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Vol. 101

COAL DIVISION

1932

PAPERS AND DISCUSSIONS PRESENTED BEFORE THE DIVISION AT THE MEETINGS HELD AT BLUEFIELD, W. VA., OCTOBER, 1931 AND AT NEW YORK, FEBRUARY, 1932

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FOREWORD

This third Coal Division volume contains papers presented at the fall meeting at Bluefield, W. Va., in October, 1931, and at the annual meeting at New York, in February, 1932.

At Bluefield the Institute was fortunate in having Prof. R. V. Wheeler, of the Department of Fuel Technology, University of Sheffield (England), present, in person, a review of the British Coal Mines Act of 1930, embracing a plan of coal marketing similar to the plan recently drawn up in this country and now being tested in the courts as to its legality.

A comprehensive résumé of the West Virginia low-volatile coals is given in one paper. Another paper sets forth the physical characteristics of various West Virginia coals.

The paper comparing mine accidents in high-volatile and low-volatile coal mines disclosed that carelessness leads to increased accidents, regardless of the field.

Fourteen papers covering the classification, use and properties of coals are included, some of which—continuations of previous papers—have progressed to the point where they are of practical use.

There are only two papers on operating problems in this volume; the papers on the growth of coal preparation of the smokeless fields of West Virginia and the recent research on subsidence of roof being the only ones in this category.

The discussion on the report of the Coal Mine Ventilating Committee at the Bluefield meeting developed the fact that there is a very urgent need for positive and specific definitions in coal mine ventilation and it is hoped that the new coal mine ventilation committee of the Coal Division will complete this code and present it for the approval of the Institute.

Thomas G. Fear,
Chairman, Coal Division, 1932.

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British Coal Industry Law Reviewed

By R. V. Wheeler,* Sheffield, England

(Bluefield Meeting, October, 1931)

A matter of much concern at present to the coal-mining industry of Great Britain is the effect of the working of the Coal Mines Act, 1930. This Act, which received the Royal Assent on Aug. 1, 1930, contains five parts: (I) production, supply and sale of coal; (II) the reorganization of the coal-mining industry; (III) hours of work; (IV) the Coal Mines National Industrial Board; (V) general. The first three parts are of major interest and discussion can be confined to them.

Part I.—Production, Supply and Sale of Coal

Under Part I of the Coal Mines Act, 1930, provision is made for central and district schemes to regulate the production, supply and sale of coal by the fixation of standard tonnage, quota and minimum prices. By Nov. 1, 1930, all the district schemes came into force with the exception of that for Scotland, where the majority of the coal-owners did not see their way to accept the amendments to their draft scheme demanded by the Mines Department. A compulsory scheme for Scotland was thereupon prepared by the Mines Department and put into force on Nov. 24, 1930. It is administered by the coal-owners of the district.

The essence of the schemes is the allocation of tonnage for each district by a Central Council, and the allocation of quota for each pit in a district by an Executive Board for that district.

Up to Dec. 31, 1930, the various bodies of coal-owners engaged in administering the schemes were employed in work preliminary to their full operation. Standard tonnages had to be determined and coals had to be classified. In consequence, it was not found possible to put the regulation of output into force until Jan. 1, 1931. The determination of minimum prices proved to be an even longer process, and only in a few districts were minimum price schedules operative during the first quarter of this year.

The working of the schemes during the March quarter, 1931, has been reviewed in an official publication by the Mines Department,[1] which can be summarized as follows:

---

* Department of Fuel Technology, University of Sheffield.

[1] Coal Mines Act, 1930. The Working of Schemes under Part I of the Act during the March Quarter, 1931. H. M. Stationery Office, Cmd. 3905, 1931.

Regulation of Output

(*a*) *The Output for the Quarter.*—The allocation for each district for the first quarter was fixed at the outset, by the Central Council, at 10 per cent below the corresponding quarter of 1930, it being anticipated that, owing to reduction in exports and depreciation of inland trade, the demand during the first quarter of 1931 would be abnormally low. This allocation was subject to adjustment in the different districts. The principle thus adopted by the Central Council was criticized as being an attempt to reduce the output of coal artificially, but the actual working of the first quarter of 1931 has shown that, for the country as a whole, the allocation was not far wrong. The output for the quarter (omitting South Wales, where a labor dispute in January complicated matters) was about one million tons below the allocation, out of a total of about fifty-one million tons.

Another criticism with respect to the regulation of output was that of inelasticity. This criticism was based mainly on the working of the scheme in the Midland (Amalgamated) district, which comprises the coal fields of Yorkshire, Nottinghamshire and North Derbyshire. In that district, the quota for each pit was fixed monthly and it became clear towards the end of February that the balance of the quarter's allocation for the whole district available for production during March would represent a quota of only about 58 per cent, which would mean much short-time working at the pits and a shortage of coal in the district. The Executive Board therefore applied to the Central Council for a further allocation for the whole district. This was refused, a number of other districts objecting on the score, disputed by those principally concerned, that the Midland coal-owners had adopted a policy of price-cutting during January and February, which had enabled them to secure business at the expense of the other districts. This firmness on the part of the Central Council, which had to take account of the position of all districts, is not regarded as evidence of lack of elasticity in the method of regulation of output, for, although there was a shortage of certain classes of coal from the Midlands during March and early April, there was no national shortage.

The various districts can be divided into two groups according as the allocation of tonnage by the Central Council was (*a*) below, or slightly in excess of, or (*b*) considerably in excess of the output as returned by the District Executive Boards. The first group includes Northumberland, Lancashire and Cheshire, Warwickshire, Cannock Chase, Midland Amalgamated, Scotland, North Staffordshire, North Wales, South Staffordshire, Kent, Forest of Dean, Somerset and Shropshire. The second group includes Cumberland, Bristol, Durham and South Wales.

In the first group, those districts (the first five) which definitely exceeded their allocations were liable, under the terms of the Central Scheme, to pay a penalty to the Central Council varying from about £75 ($375) for Lancashire and Cheshire to about £3500 ($17,500) for Northumberland. The Midland (Amalgamated) district so regulated its output in the long run that the district allocation was only exceeded by about 20,000 tons, thus becoming liable to about £2500 ($12,500) in penalties. In the second group, the output was considerably less than the allocation for various reasons. In Cumberland, an accident at the Haig pit, Whitehaven, was largely responsible. In Bristol and South Wales, there were labor disputes. In Durham, there was a shortage of trade due to foreign competition in the export market.

(*b*) *Standard Tonnages.*—The fixing of standard tonnages for the different pits by the executive boards of each district does not appear to have caused much trouble, though in most districts there were numerous appeals against the tonnages originally fixed. Many of these appeals were settled without recourse to arbitration. A much advertised exception, there were seven or eight others, was that relating to a group of pits in the Barnsley area; *i.e.*, Midland (Amalgamated) district. As a result of arbitration, the standard tonnages of the pits in question were increased.

(*c*) *Quotas.*—The fact that a district has not exceeded its allocation does not necessarily mean that no individual coal-owner in the district has exceeded his permitted output, but there does not appear to have been any serious attempt on the part of individuals to ignore the regulation of output. The occasions on which penalties can be inflicted vary according to the duration of the quota, which is sometimes fixed monthly and sometimes quarterly. In Lancashire and Cheshire, where there is a monthly quota, a few coal-owners produced slightly in excess of their permitted output in January and February. No monetary penalty was imposed, but the excess output was deducted from the succeeding quota. Individual owners in this district who exceeded their quota in March, however, were penalized 2s.6d. (62¢) per ton.

In nearly all districts advantage has been taken of the power of coal-owners to make between themselves arrangements for the transfer of quotas. The practice has varied in the different districts with regard to the price at which quota has been sold. Often enough the price has been nominal.

Price Regulation

While regulation of output, on the whole, proceeded satisfactorily during the quarter, this cannot be said for price regulation, there being, for one reason or another, considerable delay in several districts in the fixing of statutory minimum prices. The Central Council appointed a

Committee to examine interdistrict coordination of prices, and negotiations were begun between certain districts with regard to the adoption of a common basis for export and shipment prices, but there was little progress during the first quarter so far as prices for the inland markets are concerned. During the second quarter, however, a common basis for minimum prices for coal for inland markets, other than locomotive coal, and for export, bunker and coastwise shipments was reached by the Midland (Amalgamated), Cannock Chase, Forest of Dean, Shropshire, Lancashire and Cheshire, North Staffordshire, South Staffordshire, Somerset, Warwickshire and North Wales districts.

The arrangement is as follows: The basic price on April 30, 1931 is taken to be 90 per cent of the average price obtained for the various classes of coal during the period of 12 months ended June 30, 1930. The minimum price from May 1, 1931 is the basic price plus a flat rate increase of one shilling (25¢) per ton for all classes of coal to which the agreement applies (with the exception of coking slack, the basic price of which is subject to a deduction of one shilling (25¢) per ton) and further increases (*a*) of 1s. 6d. (37¢) per ton for coal for gasmaking and public utility undertakings, and (*b*) of 1s. (25¢) per ton for coal for other industrial purposes.

Sufficient experience has not yet been gained of the working of the scheme as a whole to express any opinion as to its ultimate effects on the industry, but it is to be noted that during the second quarter of the year the various schemes as regards quota provisions worked more smoothly than during the first quarter, presumably because most of the districts had in the meantime adopted minimum price schedules.

During the parliamentary debates on the Bill, certain provisions for central and district export levies were deleted. In some quarters, it is now proposed that these should be revived, either voluntarily or by legislation.

PART II.—REORGANIZATION OF THE COAL-MINING INDUSTRY

A Coal Mines Reorganization Commission has been appointed and has recently (August, 1931) issued a memorandum, outlining its policy.[2] The duty of the Commission as defined in the Act is "to further the reorganization of the coal-mining industry . . . and for that purpose to promote and assist, by the preparation of schemes and otherwise, the amalgamation of undertakings consisting of or comprising coal mines where such amalgamations appear to the Commission to be in the national interest."

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[2] Preliminary Notes on the Application of Part II of the Coal Mines Act, 1930, to the Coal Fields of Great Britain.

No voluntary amalgamations have been notified since the 1930 Act was passed, but several important schemes, including a comprehensive scheme for the whole of the Lancashire and Cheshire coal field, are reported to be under consideration. The Commission, in the memorandum referred to, suggests dividing the country into six geographical units, in each of which there should be only one colliery undertaking, as follows: Scotland; Northeast (Northumberland and Durham); Northwest (Lancashire and Cheshire); East Midlands (West Yorkshire, South Yorkshire, Nottinghamshire and Derbyshire); Central Midlands (North Staffordshire, South Staffordshire and Cannock Chase, Shropshire, Warwickshire, South Derbyshire and Leicestershire); and South Wales.

The Commission considers that the vital need of the coal-mining industry is concentration of productive effort and regards consolidation as the only sure way of bringing about such a concentration. It recognizes the probable value to the industry of sales organizations but points out that these do nothing to concentrate productive effort. The quota system it condemns as giving an artificial value to the intrinsically valueless. In support of this laudation of amalgamations, the Commission quotes the Prime Minister (Second Reading Debate on Dec. 19, 1929) with respect to the temporary character of Part I of the Act: "When you have got amalgamations, as we shall have, when royalties are nationalized, as they must be, and without delay, then the conditions which make Part I of this Bill necessary will have completely disappeared."

Part III.—Hours of Work

Provision was made for the reduction of hours of labor to a maximum of 7½, including one winding shift, to come into force four months after the passing of the Act. This interval of time was intended to provide for results to accrue from the operation of Part I of the Act. Actually, the time proved too short, but by the beginning of 1931 new agreements had been arrived at in most of the coal fields, the only serious dislocations occurring in Scotland and South Wales.

When the Coal Mines Act, 1926, lapsed, on July 8, 1931, there should have been an automatic reversion to a 7-hr. day. It was generally realized that such a change was economically impossible and Parliament, in consequence, passed a short act (the Coal Mines Act, 1931) providing for the continuance of the 7½-hr. day for a further period of one year or until the ratification of the Geneva Convention on Hours of Work in Coal Mines, if that should come earlier. By this Act, the "spread-over" of hours, which is not provided for in the Geneva Convention, is inferentially barred. The "spread-over" of hours, regarded with favor in some districts (notably Scotland and North Wales), arranged, for example, that pits which had previously worked 12 days in each fortnight would work only 11 days, the hours of work each day remaining the same.

TAXATION OF COAL MINES

There is no direct Imperial taxation of coal mines, such as is found in France and other European countries where the minerals are vested in the state. There is, however, a tax on royalties and wayleaves of 1s (25¢) in the pound (the Mineral Rights Duty). The amount paid annually in royalties is about 6d. (12¢) per ton of coal raised, or 6 million pounds (30 million dollars).

A form of taxation is the Miners' Welfare Fund, obtained by a levy of 1d. (2¢) per ton of coal raised (Mining Industry Act, 1920). In addition, 1s. (25¢) per pound of rental value is exacted from royalty owners towards the provision of pit-head baths (Mining Industry Act, 1926). The operation of the Miners' Welfare Fund has been extended for a period of five years by an Act passed in July of this year, but a Commission has been appointed to study the financial position of the fund, there being a considerable surplus in hand.

Coal mines are subject to local rates (local taxes), but the Local Government Act, 1929, provides for the partial derating (lower assessment) of coal mines. In 1928, local rates on coal mines amounted to £3,812,000 (about 19 million dollars) for the whole of Great Britain.

DISCUSSION

(*Howard N. Eavenson presiding*)

R. V. WHEELER.—Several written questions have been sent to me. Mr. Alexander asks: (1) "Is the Central Council a government or a mine-owners' organization?" It is a government council appointed by the Mines Department, and regulations are laid down as to its constitution; namely, that it shall consist partly of coal producers and partly of coal users, and that of the coal producers one-half shall be coal owners and one-half shall be miners' representatives.

(2) "Do they have any general law against consolidations or monopolies?" There is no general law against consolidations or monopolies. In fact, the committee appointed for the reorganization of the coal-mining industry definitely encourages such consolidations.

(3) "To what extent has assignment of quotas or fixing of prices closed mines?" The assignment of quotas has in some instances closed mines, because a nonproductive mine has sold its quota to a neighboring productive mine, and has itself ceased output.

Mr. Young asks: "What plan has been proposed for nationalizing royalties," and "what plan has been proposed for appraising the plants of what would be active and inactive mines under the amalgamation?" No plans have yet been made public for the nationalization of royalties or for the appraising of plants under any amalgamation schemes.

Mr. Holmes asks: "Does the government establish definite zones wherein each coal field can market its product locally, or are freight rates relied upon to govern this situation?" There is no restriction on the sale of coal in one district that has been produced in another district; freight rates are relied upon for control.

J. T. RYAN, Pittsburgh, Pa.—I visited the British coal fields recently, and was interested in studying the workings of the Coal Mines Act, and I confess I was greatly puzzled in drawing a picture of it, because the information was most confusing. I want to compliment Professor Wheeler for the clear manner in which he has pictured the working of this Act in Great Britain.

Most of our coal people will say immediately that we cannot do the things in this country that have been done in Great Britain. We may not be able to do them in the same way, but certain definite things have been accomplished by this Coal Mines Act.

I think that Professor Wheeler will agree that the consensus of opinion is that it is not working out very well. I think the depression is being confused with the workings of this Act.

Three very specific things have been accomplished that we will have to accomplish some day in our coal industry: (1) By the workings of this Act a minimum selling price for coal has been established; (2) a stabilized wage rate has been established; and (3) a production quota in line with the demand has been established.

While many other details have not worked out so well, those three things that are so vital to the coal industry in Great Britain have been accomplished.

There is one complication that in Great Britain that I do not think we have here; that is, their export business. They cannot control it as they can their domestic business. We would not have that complication, because the quantity of our coal exported is almost negligible.

J. H. **Pierce**, New York, N. Y.—As I recall, about two years ago there were voluntary associations in Scotland that contributed to a fund to subsidize exports. Did the Coal Mines Act affect the agreements then in existence or attempt to set up a differential?

R. V. **Wheeler**.—The Government wiped out the agreements altogether. The owners in some districts had perfectly sound schemes of their own which were working well. As regards an arrangement for export levies, that has been wiped out by the Act of 1930, but in some quarters there is a desire to have it re-introduced.

A. B. **Crichton**, Johnstown, Pa.—In the United States we have been greatly interested in this subject. I have been especially interested to learn how operation of the Coal Mines Act was being carried out.

Stabilization of the coal industry through government regulation has not generally been approved, so that any similar act thought of in this country should include the very least possible government interference to find approval in the coal industry. I question whether we can allocate the tonnage and regulate the output of the different mines in this country as easily as in Great Britain, because of the more limited field there. As I understand the paper, the mines of the country have been divided into six different districts, and even with the limited number of mines as compared with this country, there has been a good deal of trouble in carrying out the scheme.

I do not see how we could gain much in this country unless we also limited the development of additional coal property and controlled the opening up of mines now abandoned.

I understand that price-fixing is not legal in this country under our present laws, but the minimum price for coal seems a reasonable feature. There ought not to be any objection to giving a living wage to the coal miners, and if a minimum price for coal were fixed at the cost of production at the lowest cost mines, it would seem to be a fair provision.

One feature of importance has not been mentioned here or in any of the other meetings I have attended, although it has been urged upon the country by the United Mine Workers at various times; that is, the question of limiting the hours of labor per day and limiting the number of days per week. It is one of the most important things we have to consider, and if that one thing could be done by statutory regulation, in my opinion much further regulation of the industry would not be necessary. Hours of labor in industry have been reduced greatly over the years by statutory regulation, and it should be continued. It would tend to regulate the peaks in industry better than anything else. With the plenteous supply of every product, still the country is suffering, economists tell us, because of an unequal distribution of wealth. Limiting the hours of labor would certainly help to a better distribution of working time as applied to natural resources, particularly the coal industry.

Mention is also made of the method of taxing coal lands, which apparently is high in Great Britain, and I assume from Professor Wheeler's statements that they are trying to cut down taxes and limit them largely to a production tax. We would be much better off in this country if we could have a nominal tax on undeveloped coal land, with a production tax on coal as shipped. There would not be the incentive to open up undeveloped coal lands, because of the heavy carrying charges, as at present.

I would not be understood as favoring government regulation of coal mines in this country—it ought to be done from within the industry—but I can see no harm involved in limiting the hours of labor in coal mines.

G. S. Patterson, Huntington, W. Va.—How are sales of coal handled in Great Britain? We have coal salesmen traveling over the country, numbers of them, six or a dozen of them calling on one man in a day.

R. V. Wheeler.—We have the same method. There is no general sales organization. The sales are made by coal factors who usually sell the coal on its name rather than on its proved quality.

H. N. Eavenson, Pittsburgh, Pa.—It is said there are no taxes on undeveloped coal land in England. Is that true?

R. V. Wheeler.—Yes.

J. L. Steinbugler, New York, N. Y.—What provision exists in the Mines Act for the collection of the minimum price fixed from time to time? How is the collection made obligatory on an owner who would prefer to sell at a lower price to maintain his quota?

R. V. Wheeler.—The Central Committee has the power to impose and recover from the owner monetary penalties for failure to comply with the scheme. I do not know how they exercise that power.

E. L. Greever, Tazewell, Va.—I know of no question more difficult under the American system of laws than the one before us. We have a dual system of government which makes it extremely difficult, because the federal government controls interstate commerce and the state government has control over the property.

We have only one really fundamental thing to meet and that is the question of production. The overproductive capacity of the coal mines of this country is the sole cause of our troubles. They tell us the coal men can set their own goals and solve their own problems and settle their own difficulties. I agree with that as a principle; but when the coal men undertake to settle their own problems that means cooperation. Cooperation means agreements. Agreements, to be effective and reach the trouble, must be agreements to control the production, and there we run right into the Sherman Act. That is the problem as I see it, and the answer would seem to be some sort of revision of the Sherman Act. There should be some sort of relaxation of the rigid requirements of that Act that would afford some way to get away from the interpretation placed on the Act by the Courts.

It would seem that the federal government is helpless when it undertakes to control production. Our Supreme Court has said more than once that the production of coal has nothing to do with interstate commerce, and the result of all the thought I have been able to give to the subject is that so far as federal law is concerned we already have too much. Instead of having more regulatory laws we ought to have a repeal or modification of laws which restrict men from entering into contracts that are sound economically, and which must be entered into if this industry is to prosper.

As I understand it, in England there is no such restriction on the government; therefore, the Act of that country has possibilities there, while such an act cannot even be considered here.

G. S. Rice, Washington, D. C.—In recent years it is constantly suggested that the Sherman Act stands in the way of stabilizing the coal industry. My observations on the coal-mining industry go back nearly 44 years, 21 of which were in the engineering and operating end of coal mining, nearly all of the latter experience having been before the Sherman Act came into effect. Therefore I know at first hand that before it was enacted the coal-operating business constantly had serious troubles.

In 1891 I went to mines in Iowa and Illinois. In the next few years troubles were brewing in the bituminous industry. There was severe unrestricted competition which led to lowering prices; only the mining companies that were favored by the powerful railway lines carried on successfully. This was in contrast with the impoverished condition of unfavored operators, which was one of the reasons pressure was brought by the public for establishing the Sherman Act. Prior to its enactment there was extreme competition in selling in many districts, notably in Illinois. Mine operators were selling coal at ridiculously low prices. Some run-of-mine coal was sold at as low a price as 35¢ per ton on the car. They could not get together and agree. This caused lowering of wages to the starvation point and led to strikes and riots, then to legislative action in many states and the springing up of the United Mine Workers into power.

I do not believe that you will get what you want, unless human beings have changed, by simply removing the Sherman Act. Something radically new must be worked out for stabilization.

E. L. Greever.—I only suggested that would give a possibility.

M. D. Cooper, Pittsburgh, Pa.—We want to ask Dr. Wheeler how the production at the present time under the Act compares with the previous normal years.

R. V. Wheeler.—So far as the first quarter is concerned, it is about one-half million tons under the estimate. But our annual production of coal is about what it has been.

J. L. Steinbugler.—Under the 1930 Act, is the owner of undeveloped coal land free to develop it at any time?

R. V. Wheeler.—Yes.

J. L. Steinbugler.—How would the tonnage be determined at the beginning?

R. V. Wheeler.—By agreement with those concerned, through the local Standard Tonnage Committee and, if necessary, by arbitration.

J. L. Steinbugler.—How often is the quota determined for each colliery?

R. V. Wheeler.—Every quarter.

H. F. Bain, New York, N. Y.—They used to work under the quota system in the nitrate fields of Chile. Any colliery would lose its quota if it did not work a certain number of weeks per year, so they would fire up and work several weeks in order not to lose it and then sell it for the rest of the year.

Comparative Frequency of Accidents in High-volatile and Low-volatile Coal Fields of West Virginia

By R. F. Roth, Fairmont, W. Va.

(Bluefield Meeting, October, 1931)

"Are mine accidents more frequent in low-volatile than in high-volatile mines?" is a question often asked, frequently discussed and in most instances answerable in the affirmative from only a casual inspection of the reports at hand. Most conclusions as to their relative frequency as well as to the relative frequency of the several classes of accidents in these coal fields have been more superficial than accurately informative, the exceptions, of course, being the reports of the few investigators who have gone to the trouble to make a detailed analysis of the subject.

The purpose of this paper is to endeavor to answer the question as pertains to the coal fields of West Virginia. It is not intended as criticism nor as an indictment of any particular group of operators for any unfavorable differences that may be shown to exist. It is intended merely as an impartial analysis of accident frequency, with brief references to possible causes and effect, and is presented with the hope that it may prove informative, at least in part, to all concerned and helpful in particular to those engaged in the solution of the perplexing problems of safety and accident prevention in the coal mines of West Virginia.

The comparison was developed with 12 graphs covering the 10-year period from 1921 to 1930, plotted from data furnished by the State Department of Mines. Low-volatile territory was taken to include Fayette, Greenbrier, McDowell, Mercer, Raleigh, Webster, and Wyoming counties; high-volatile territory to include all other counties of the state.

The "accident frequency" figures shown on the graphs represent only "normal" accidents, by which are meant accidents other than major disasters such as general explosions of gas and dust wherein the fatalities exceeded four persons.

The frequency of such major disasters was the same in both fields (six in number) during the 10-year period of 1921 to 1930, and to include them would add nothing of value; moreover, they would cause confusing distortions on the graphs at their particular intervals of occurrence.

It may be interesting, however, to note in passing that the severity of general explosions has been greatest in the high-volatile field, where 281 persons have been killed as against 140 in the low-volatile field.

The classification of accidents is shown by the graphs of Figs. 1 to 12. The graphs of Figs. 1 to 9 are based on a common datum of "accidents per million tons mined"; those of Figs. 10 to 12 are based on a straight percentage of the total accidents of their respective classes.

Total Fatal Accidents (Fig. 1)

It is evident by inspection that the low-volatile field carries a higher frequency rate than the high-volatile and that the difference of 1.4 fatal accidents per million tons mined has not only been persistent throughout the 10-year period but was considerably greater during the years 1924, 1925, 1926 and 1929. The general trend in both fields has been downward but the sharp peaks along the low-volatile line indicate a more erratic occurrence than in the high-volatile field. The average frequency for the 10-year period in both fields, and in the entire bituminous industry of the United States, is shown in Table 1.

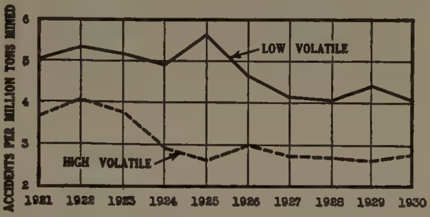

Fig. 1.—Total fatal accidents.

Table 1.—*Fatal Accidents*

Per Million Tons Mined

	Average, 10-year Period, 1921–1930	Year 1930
Low-volatile field, West Virginia	4.68	4.06
High-volatile field, West Virginia	3.07	2.75
Entire state, West Virginia	3.76	3.36
Bituminous fields, United States	3.53	3.41

In the low-volatile field the frequency rate has been below the 1921 level ever since 1924 with the exception of the year 1925, and the sharp peak during that year no doubt was partly due to the fact that the 1925 report included the 18 months from July 1, 1924 to Dec. 31, 1925, in accordance with an amendment to the state mining laws.

It is most gratifying to note that in 1930 the frequency rate for the entire state of West Virginia was lower than the rate for the entire bituminous industry, and that the reduced figures are applicable to more than one-fourth of the tonnage mined in the combined coal fields of the whole United States.

TOTAL NONFATAL ACCIDENTS (FIG. 2)

In this division, also, there is a higher frequency rate in the low-volatile field. Beginning in 1921, with a difference of 5 accidents, both fields run along with almost parallel lines until the year 1924, when a marked increase is noticeable in the low-volatile field, and this continues until the year 1927, when the difference between the two fields has reached a total of 17 accidents, a gain of 240 per cent. At the end of 1928 the difference had mounted to 23 accidents per million tons mined, an increase of 360 per cent over the 1921 frequency rate.

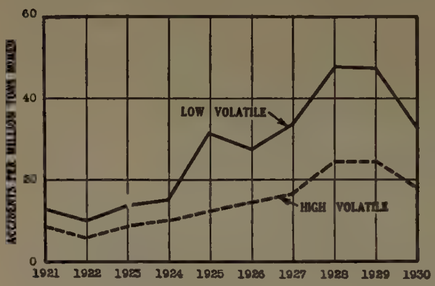

FIG. 2.—TOTAL NONFATAL ACCIDENTS.

The general trend in both fields was sharply upward until the year 1928 when it suddenly stopped, leveled off across 1929 and then took a sharp trend downward. Is the 1928–1929 connecting line the summit, and is the decline through and beyond the year 1930 to continue until the nonfatal accident frequency has reached new low levels throughout the state? Time alone can tell the story.

TOTAL ACCIDENTS—FATAL AND NONFATAL (FIG. 3)

As a composite of graphs 1 and 2 this develops nothing of particular value, but it is interesting to note its similarity to Fig. 2. It shows in no uncertain manner the predominating frequency of the nonfatal accidents.

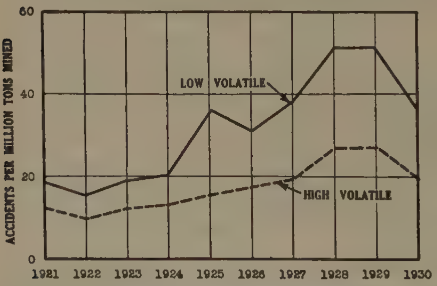

FIG. 3.—TOTAL ACCIDENTS, FATAL AND NONFATAL.

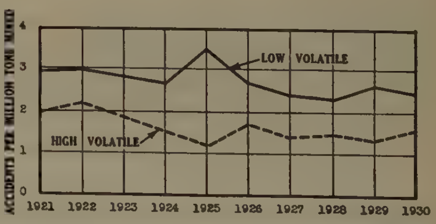

FIG. 4.—FATAL ACCIDENTS FROM FALLS OF ROOF AND COAL.

FATAL ACCIDENTS FROM FALLS OF ROOF AND COAL (FIG. 4)

The difference in frequency rates in this class of fatal accidents is again favorable to the high-volatile field. Beginning with a difference of practically one accident per million tons mined, in 1921 both fields begin a trek, principally downward and quite uniform throughout.

The frequency rates, as can readily be seen on the chart, do not vary more than a fractional part of one accident, excepting that in the year 1925 the low-volatile rate jumped 0.8 of an accident while the high-volatile field reached its lowest level after a steady decline since the year 1922 and then jumped 0.5 of an accident in the year 1926.

It is well to note here that falls of roof and coal are responsible for 56 per cent of all the normal fatal accidents in the coal mines of West Virginia during the past 10 years, and that the low-volatile field is charged with 31 per cent and the high-volatile with 25 per cent of the same total. A comparison of frequency rates is given in Table 2.

TABLE 2.—*Fatal Accidents from Falls of Roof or Coal, West Virginia*
Per Million Tons Mined

	Average 10-year Period, 1921–1930	Year, 1921	Year, 1930
Low-volatile field	2.78	2.96	2.50
High-volatile field	1.63	2.02	1.58
Entire state	2.10	2.42	2.00

NONFATAL ACCIDENTS FROM FALLS OF ROOF AND COAL (FIG. 5)

This graph shows a general upward trend of the frequency of this class of nonfatal accidents in both fields during the 10-year period 1921 to 1930. During the year 1929 there was a marked decrease and then a decline through the year 1930.

There is a striking similarity between this graph and that in Fig. 2, and the influence of this group upon the total nonfatal accidents can be realized almost at a glance. The actual figures for this class of accidents show 20 per cent of the total number of nonfatal accidents in the low-volatile field, 11 per cent in the high-volatile field, and 31 per cent in the entire state of West Virginia. In outstanding contrast with the fatal accidents from the same cause, which rank first in their group, this class of accidents ranks third and therefore the lowest in its group.

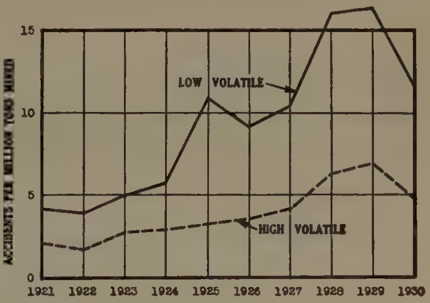

FIG. 5.—NONFATAL ACCIDENTS FROM FALLS OF ROOF AND COAL.

The increase of frequency was more rapid in the low-volatile field from 1922 to 1928 than in the high-volatile and likewise its decline has been more rapid since 1929. The difference in frequency between the two fields increased from 2.1 accidents per million tons mined in 1921 to 6.74 in 1930, all in favor of the high-volatile field. A comparison of frequency figures is shown in Table 3.

TABLE 3.—*Nonfatal Accidents from Falls of Roof and Coal*
Per Million Tons Mined

	Average, 10-year Period, 1921–1930	Year, 1921	Year, 1930
Low-volatile field	9.36	4.27	11.72
High-volatile field	3.85	2.15	4.98
State of West Virginia	6.15	3.05	8.11

FATAL ACCIDENTS FROM MINE CARS AND MOTORS (FIG. 6)

The general trend of accident frequency in this class has been visibly downward in both fields during the past 10 years and at no time since 1922 has the frequency rate in either field reached or exceeded the level of that year. The difference in frequency between the two fields is again favorable to the high-volatile zone. In 1921 it was 0.48 of an accident per million tons mined and in 1930 it was 0.39 of an accident. The variation of frequency difference has been very slight during the 10-year period excepting in the years 1923 and 1928, when a trend in opposite directions reversed the advantage in the first instance and reduced it approximately one-half in the second instance. Frequency rates are given in Table 4.

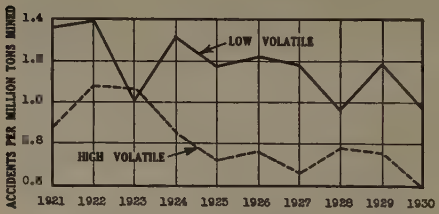

FIG. 6.—FATAL ACCIDENTS FROM MINE CARS AND MOTORS.

TABLE 4.—*Fatal Accidents from Mine Cars and Motors*
Per Million Tons Mined

	Average, 10-year Period, 1921–1930	Year, 1921	Year, 1930
Low-volatile field	1.18	1.36	0.98
High-volatile field	0.81	0.88	0.59
State of West Virginia	0.97	1.09	0.77

This class of accidents ranks second to falls of roof and coal in the percentage of total fatal accidents, there having been 13 per cent in the low-volatile field, 12 per cent in the high-volatile field, and 25 per cent in the entire state of West Virginia.

NONFATAL ACCIDENTS FROM MINE CARS AND MOTORS (FIG. 7)

The frequency characteristics of this graph are similar to those of No. 5, which depicts the nonfatal accidents from falls of roof and coal. The upward trend in the high-volatile field after the year 1922 is more rapid and also more uniform for the mine-car and motor accidents than it is for the same field in the other class of nonfatal accidents.

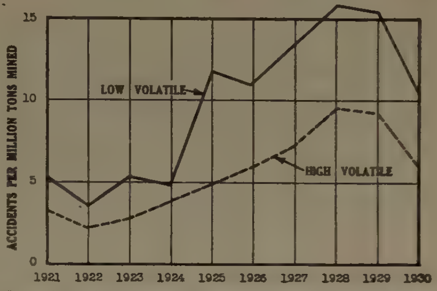

FIG. 7.—NONFATAL ACCIDENTS FROM MINE CARS AND MOTORS.

The difference in frequency is once more in favor of the high-volatile field and has increased during the 10-year period from 2.28 accidents in 1921 to 4.53 accidents in 1930. The gap between the two fields has been considerably more than doubled on an average across the 10-year period. Marked increases of accident frequency prevail in both fields during the 10-year period, as shown in Table 5.

TABLE 5.—*Nonfatal Accidents from Mine Cars and Motors*
Per Million Tons Mined

	Average, 10-year Period, 1921–1930	Year, 1921	Year, 1930
Low-volatile field	9.73	5.49	10.40
High-volatile field	5.47	8.21	5.87
State of West Virginia	7.24	4.18	8.00

This class of accidents ranks highest of the nonfatal group, the percentages of the total number of nonfatal accidents being 21 per cent in the low-volatile field, 16 per cent in the high-volatile field, and 37 per cent in the entire state of West Virginia.

FATAL ACCIDENTS FROM ALL OTHER CAUSES (FIG. 8)

The frequency rate of this class of accidents shows a general downward trend in both fields since the year 1923. Beginning in 1921 with a frequency difference of only 0.03 of an accident in favor of the low-volatile field, the advantage was switched during 1922 to the high-volatile field. It continued to widen during 1923, then decreased slightly in 1924, considerably more in 1925, remained nearly the same in 1926, was switched back to the low-volatile field in 1927, was regained to a marked degree by the high-volatile in 1928 and then lessened rapidly through 1929 and 1930. At the end of 1930 both fields show exactly the same rate of accident frequency; namely, 0.8 of one accident.

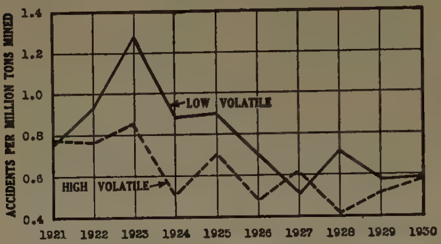

FIG. 8.—FATAL ACCIDENTS FROM ALL OTHER CAUSES.

Electricity was responsible for 30 per cent of this class of accidents during the 10-year period, mining machines were responsible for 13.5 per cent and tipples were responsible for 1.5 per cent. Comparative figures are given in Table 6.

TABLE 6.—*Percentage of Fatal Accidents from Electricity, Mining Machines and Tipples of Total Fatal Accidents from All Other Causes*

	Electricity, Per Cent	Mining Machines, Per Cent	Tipples, Per Cent	Total, Per Cent
Low-volatile field	14	5.7	0.2	19.9
High-volatile field	16	7.8	1.3	25.1
State of West Virginia	30	13.5	1.5	45.0

This group ranks third, and lowest, in percentages of total fatal accidents, with 9 per cent in the low-volatile field, 10 per cent in the high-volatile field and 19 per cent in the entire state.

NONFATAL ACCIDENTS FROM ALL OTHER CAUSES (FIG. 9)

Here again the general trend is upward in both fields until the year 1928, when the increase suddenly stopped, eased off slightly through 1929 and then began a sharp downward trend through the year 1930. The increase of frequency was considerably more pronounced in the low-volatile field from the year 1924 until 1928 than in the high-volatile.

The difference in accident frequency has been persistently in favor of the high-volatile field and increased very materially from 1924 to 1928. It has varied during the 10-year period from 0.49 of an accident per million tons

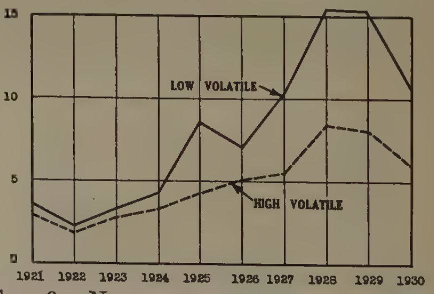

FIG. 9.—NONFATAL ACCIDENTS FROM ALL OTHER CAUSES.

mined in 1921 to 4.34 in 1930. Table 7 gives the frequency rates. Table 8 gives the percentages chargeable to electricity, mining machines and tipples.

Table 7.—*Nonfatal Accidents from Causes Not Previously Listed*
Per Million Tons Mined

	Average, 10-year Period, 1921–1930	Year, 1921	Year, 1930
Low-volatile field	8.03	3.46	10.42
High-volatile field	4.83	2.97	6.08
State of West Virginia	6.18	3.18	8.09

Table 8.—*Percentage of Nonfatal Accidents from Electricity, Mining Machines and Tipples of Total Nonfatal Accidents from All Other Causes*

	Electricity, Per Cent	Mining Machines, Per Cent	Tipples, Per Cent	Total Per Cent
Low-volatile field	2	8.0	1.0	11
High-volatile field	1	9.3	0.7	11
State of West Virginia	3	17.3	1.7	22

This class ranks second in the nonfatal accident group, with 18 per cent of all nonfatal accidents in the low-volatile field, 14 per cent in the high-volatile field, and 32 per cent in the entire state.

Comparison by Years, Percentage Total Fatal Accidents of Total Accidents (Fig. 10)

This graph shows a general downward trend of fatal accident percentages of the total for all classes, fatal and nonfatal, down through the years 1922 to 1929. The general trend of production, of course, has been upward during the same period. It has been more rapid in the high-volatile field and is accountable, at least mathematically, for the better showing in accident frequency in that field. On the other hand, it may be physically possible for the low-volatile field to increase production without increasing its percentage of fatal accidents, and perhaps not impossible to increase it in the face of a still further decline in fatalities.

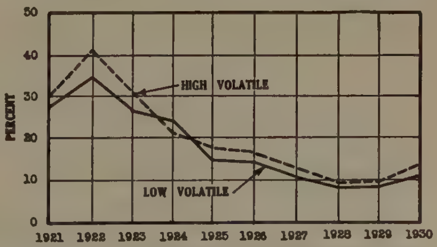

Fig. 10.—Comparison by years. Percentage of fatal accidents in total number of accidents.

It will be noticed that the graph lines for the two fields have gone along quite uniformly through the 10-year period with the exception of the years 1924 and 1925, and that the low-volatile field enjoys somewhat lower percentages throughout most of the period.

COMPARISON BY YEARS. PERCENTAGE TOTAL ROOF ACCIDENTS OF TOTAL ACCIDENTS (FIG. 11)

This graph shows that both fields have gone along quite uniformly through the 10-year period with very little change in their yearly percentages of all classes of roof accidents of total accidents and that the frequency differences can again be accounted for at least mathematically by the difference in the trend of production.

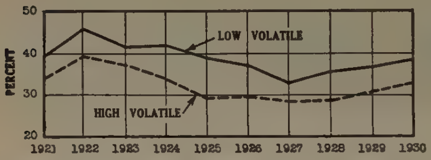

FIG. 11.—COMPARISON BY YEARS. PERCENTAGE OF ALL ACCIDENTS CAUSED BY ROOF FALLS.

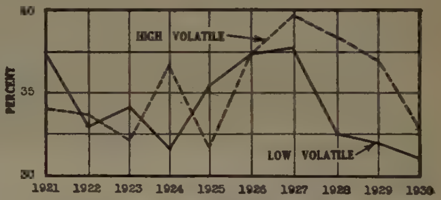

FIG. 12.—COMPARISON BY YEARS. PERCENTAGE OF MINE-CAR AND MOTOR ACCIDENTS IN TOTAL NUMBER OF ACCIDENTS.

COMPARISON BY YEARS. PERCENTAGE TOTAL MINE-CAR AND MOTOR ACCIDENTS OF TOTAL ACCIDENTS (FIG. 12)

The yearly percentages of all mine-car and motor accidents have varied but little during the 10-year period and the favorable percentages have been enjoyed almost equally by both fields. This graph shows plainly that regardless of an upward production trend the favorable percentages of accidents of any particular class may be held by either field, and that influences other than tonnage must have had a powerful bearing on the switch from one field to the other.

GENERAL TRENDS

The general trend of fatal accident frequency in the mines of West Virginia during the 10-year period 1921 to 1930 has been favorably downward.

The general trend of nonfatal accident frequency for the same period has been decidedly upward.

The general trend of frequency of total accidents, fatal and nonfatal, has been upward, because of the overshadowing influence of the nonfatal class.

The general trend of fatal accidents of all kinds in the United States has been appallingly upward during the 10-year period. A compilation of graph and statistical figures from data furnished by the Metropolitan Life Insurance Co. tells the story in detail (Fig. 13). We are told that 16,000 industrial workers were killed throughout the United States during the year 1930. The contribution from West Virginia's coal mines to this staggering total was 412 fatalities, or 2.5 per cent.

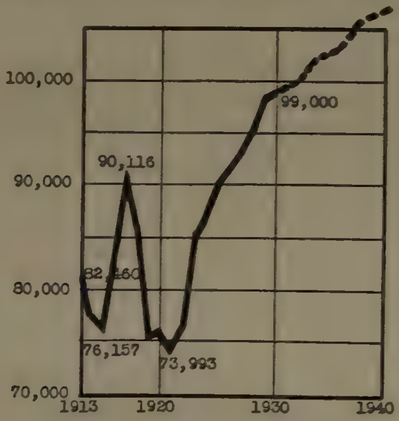

FIG. 13.—ACCIDENTAL DEATHS IN THE UNITED STATES.
33,000 killed in motor-vehicle accidents during 1930.
30,000 accidentally killed in homes during 1930.
20,000 fatal accidents in public places during 1930.
16,000 industrial workers accidentally killed during 1930.

This graph and the statistics are taken from a leaflet entitled The Rising Toll of Accidents, distributed by the Metropolitan Life Insurance Company.

CAUSES OF VARIATION IN WEST VIRGINIA FIELDS

Now what is there to say regarding the unfavorable difference of accident frequency in the low-volatile as against the high-volatile coal fields of West Virginia?

1. Would a comparison by millions of man-hours instead of millions of tons mined show the situation in a different light?

2. Is the productivity less in the low-volatile field, and could it be increased without increasing the accident rates, so that a greater production would lower the accident frequency rates as compared with the high-volatile field?

3. Has nature been more unkind to the low-volatile area with regard to roof and grades, on account of different geological formations; for instance, the estuary theory versus the peat-bog theory?

4. Has metamorphism placed a handicap on the low-volatile field in the matter of roof control, by the well-known friable nature of the coal?

5. Have accidents increased comparatively in number in low-volatile mines wherein first mining has been completed without pillaring and wherein the pillars are now being drawn throughout, and what is the increase of this class of mining throughout the field?

6. How does the low-volatile field differ from the high-volatile in this respect?

7. What influence does the gradual trend toward deeper mining in the low-volatile field exert on all classes of mine accidents?

8. What is the comparative accident frequency and severity in the smaller or financially weaker mines, as compared with the larger or financially stronger mines in both fields?

9. Is there any appreciable difference in accident frequency in those sections of the low-volatile field wherein seams with good roof have been exhausted and the mining transferred into seams with bad roof conditions?

10. Has one field suffered more than the other from accidents that happened in the zones traversed by underground streams or saturated from surface streams?

11. How much is known regarding the influence of improper systems of mining on all classes of mine accidents in both fields?

These questions have not been asked with the idea that they can be answered here. They are only a few of many which could be asked to illustrate the need of more intensive as well as more detailed research in connection with the humane work of mine safety and accident prevention.

What is the coal industry in West Virginia going to do about it?

The state boasts of safety organizations second to none in the mining world and no one in possession of the facts has aught but praise for their accomplishment. Their scope, however, is only too limited and should be augmented by a Bureau of Accident Research. Whether such a Bureau should function outside of the industry or whether it should be organized, financed and controlled by the operators within the industry itself is, of course, a matter which the interested parties would determine for themselves. At any rate, let it be remembered that as little as 1/40 cent levied against 100,000,000 tons production would provide the sum of $25,000 annually toward such an undertaking.

Would such an organization pay? The question might be partly answered by quoting from the July number of *Coal Mining*, as follows:

Do Safety Organizations Pay in Mining?

> This is difficult to answer with any great degree of definiteness but the following data taken from a paper by J. T. Ryan, in *Mining and Metallurgy* of October, 1930, give at least some ground for answering yes in so far as prevention of accidents in Pennsylvania coal mines is concerned.
>
> To get some real data on whether organized accident prevention work pays, the companies having an organized safety department in some fairly effective and organized form were segregated in this report; and this list included a number of companies whose safety organization is known to be ineffective due to improper organization or misplaced or divided responsibility, but which were nevertheless included to obtain the proper average.

The following comparison table gives the answer:

	Production, Tons	Number of Fatal Accidents	Nonfatal 60-Day Disability	Tonnage Produced	
				For each Fatality	For each 60-day Disability
For all Pennsylvania bituminous coal mines	142,351,359	376	2,334	378,594	60,990
Companies having organized safety departments	73,244,413	144	1,078	508,642	67,945
Companies not having organized safety departments	69,106,946	232	1,256	297,875	55,021
Selected group having well organized safety department	59,438,126	100	808	594,381	73,562

It certainly pays in other lines of endeavor; for instance, in the life insurance business, as told by the Metropolitan Life Insurance Co. in an illustrated pamphlet recently distributed by its Welfare Division, following an 18-year campaign to lengthen the span of life of its policyholders.

WHAT HAS BEEN DONE

The average Metropolitan Industrial policyholder born last year could expect to live nine years and two months longer than could a policyholder born in 1911.

In 1911 the death rate for Industrial policyholders was 24.3 per cent higher than the rate for the population of the United States as a whole. In 1929 the death rate was virtually the same for both groups. (The 1930 statistics for the general population are not yet available.)

Between 1912 and 1929, it is estimated that the lives of 398,250 Industrial policyholders have been saved over and above the saving expected from the mortality improvement in the general population during the same period.

HOW IT HAS BEEN DONE

This work of life-saving has been many-sided. A bedside nursing service for policyholders is available in 5,198 towns and cities of the United States and Canada. More than six hundred million popular health pamphlets have been distributed. Four health films have been produced and shown to nineteen million people. Cooperation has been given to social agencies and to schools, and special literature printed for their use. The company has been closely affiliated with health departments in efforts to obtain increased appropriations for health work and to gain the support of policyholders for better health laws. Metropolitan managers and agents have assisted in campaigns to secure pure water and pure milk, and to prevent the spread of epidemics. They have aided campaigns of health departments and private health agencies to prevent tuberculosis, diphtheria and smallpox, and have cooperated in other preventive health measures. Demonstrations and *research*[1] in the field of disease prevention have been financed. The closing pages of this book suggest how this work of sickness prevention and life-saving has increased in magnitude from small beginnings.

Might not a similar campaign of systematized effort and research do as much for the coal industry of West Virginia?

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[1] Italics by the author.

[For discussion of this paper, see page 42.]

Comparative Frequency of Mine Accidents in Low-volatile and High-volatile Bituminous Mines of Pennsylvania

By Rush N. Hosler,* Harrisburg, Pa.

(Bluefield Meeting, October, 1931)

The question asked in the subject of this paper may also be stated as follows: What is the relation of volatile matter in coals to accident frequency in mining them?

The reason for omitting the low-volatile coals of the Anthracite Region from consideration in this study is that its geological conditions are so different from those of the bituminous fields that the methods of mining the two coals are not analogous.

Dr. George H. Ashley, State Geologist of Pennsylvania, suggested that the Bituminous Region be divided as follows:

Low-volatile District
Counties

- Bedford
- Blair
- Cambria
- Centre
- Clearfield
- Clinton
- Huntingdon
- Lycoming
- Somerset
- Tioga

High-volatile District
Counties

- Allegheny
- Armstrong
- Beaver
- Butler
- Clarion
- Elk
- Fayette
- Greene
- Indiana
- Jefferson
- Lawrence
- Mercer
- Washington
- Westmoreland
- Venango

There may be some little overlapping of low-volatile and high-volatile coals on the county division lines, but any other dividing lines would involve a division by individual mines, which would incur such a great amount of work that it could not be undertaken at this time.

The figures used in presenting the information are taken from the Compensation Insurance Experience in Bituminous Mining in Pennsylvania and compiled by the Coal Mine Section of the Pennsylvania Compensation Rating and Inspection Bureau, Harrisburg, Pa. This experience represents a little more than 50 per cent of the total bituminous tonnage produced in Pennsylvania and an exposure of approximately 275 million tons of coal.

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* Superintendent, Coal Mine Section, Pennsylvania Compensation Rating and Inspection Bureau.

In compiling this experience, it was necessary to segregate it by counties. Table 1 shows the accident severity rate and insurance experience in the low-volatile district in 1925 to 1928, inclusive, and Table 2 brings out the same information regarding the high-volatile district for the same period. This four years' experience was used because it is the latest complete information available. The number of accidents used are those compensated under the Workmen's Compensation Act.

These accidents are divided into two classes: (1) deaths and permanent totals and (2) all others. Permanent totals are grouped with the deaths for the reason that economically they are the same as though death did occur, and cost more than the average death. The number of each class of accidents is shown on the tables in columns 3 and 4, respectively. The accident rates for these same classes are shown in columns 5 and 6. The lost time in days is reckoned in accordance with the International Industrial Accident Commission Standard Weights for Lost Time Accidents. Column 10, Days Lost per Million Tons, forms an accurate unit of measure and is the only complete unit of measure that is comparable with other states.

Following the figures on these tables, we find the accident rate per million tons for deaths and permanent totals is 3.45 and the rate for all others is 112.47, with 31,391 lost days in low-volatile coal. The corresponding figures for high-volatile coal are 3.10, 99.41 and 28,948. This shows a slightly lower rate for the high-volatile coal. Another factor should be taken into account, which would modify the figures for high-volatile coal. The ratio of production for each 250-day employee between the low-volatile and high-volatile districts is 0.87 to 1.00. Another way to state this is that in the low-volatile district each 250-day employee produces 945 tons of coal, while in the high-volatile district he produces 1086 tons. The figures 3.10 for deaths and permanents, 99.41 for all others and 28,948 lost days should be modified in accordance with this ratio in order that the comparison may be on the same exposure basis. Doing this would change these figures to 3.56 for deaths and permanents, 114.26 for all others and 33,273 lost days. These rates are slightly higher than those for low-volatile coal.

In these comparisons only accidents not involved in catastrophes have been used. (A catastrophe is defined as an accident in which five or more persons are killed.) Catastrophes should be studied separately from the other accidents, because to have a proper exposure they should include a much longer period of time. In making comparisons of catastrophes, the entire experience since the inception of the Workmen's Compensation Act, 1916 to 1930 inclusive, a 15-year period has been used, except one accident in which six were killed, which occurred

TABLE 1.—*Accident Severity Rate and Insurance Experience in Low-volatile Coal in Pennsylvania, 1925 to 1928, Inclusive*

County	Tonnage	Compensated Accidents[a]		Accident Rates per Million Tons		Total Cost[b]	Cost per Million Tons	Total Days Lost	Days Lost per Million Tons
		Deaths and Permanent Totals	All Others	Deaths and Permanent Totals	All Others				
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)
Total	89,052,360	307	10,016	3.45	112.47	$2,708,144	$30,411	2,795,399	31,391
Bedford	1,660,584	6	212	3.61	127.67	68,659	41,346	64,419	38,793
Blair	1,085,497	5	105	4.61	96.73	34,048	31,366	39,963	36,815
Cambria	38,076,132	129	4,471	3.39	117.42	1,127,947	29,623	1,198,937	31,488
Centre	2,650,979	3	248	1.13	93.55	45,737	17,253	36,447	13,749
Clearfield	13,938,338	52	1,729	3.73	124.05	453,622	32,545	469,863	33,710
Clinton	930,880	3	97	3.22	104.20	28,167	30,258	24,140	25,932
Huntingdon	577,055	2	84	3.47	145.57	30,350	52,595	20,028	34,707
Lycoming	101,736	0	9	....	88.46	1,630	16,022	708	6,959
Somerset	29,806,734	107	3,006	3.59	100.85	913,883	30,660	938,529	31,487
Tioga	224,425	0	55	....	245.07	4,101	18,273	2,365	10,538

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

TABLE 2.—*Accident Severity Rate and Insurance Experience in High-volatile Coal in Pennsylvania, 1925 to 1928, Inclusive*

County (1)	Tonnage (2)	Compensated Accidents[a] Deaths and Permanent Totals (3)	Compensated Accidents[a] All Others (4)	Accident Rates per Million Tons Deaths and Permanent Totals (5)	Accident Rates per Million Tons All Others (6)	Total Cost[b] (7)	Cost per Million Tons (8)	Total Days Lost (9)	Days Lost per Million Tons (10)
Total	184,064,732	570	18,297	3.10	99.41	$5,166,920	$ 28,071	5,328,222	28,948
Allegheny	23,713,171	71	2,055	2.99	86.66	706,126	29,778	699,814	29,512
Armstrong	9,125,581	26	991	2.85	108.60	261,082	28,610	262,616	28,778
Beaver	382,056	5	109	13.09	285.30	42,300	110,717	41,466	108,534
Butler	7,475,345	30	966	4.01	129.22	310,524	41,540	284,794	38,098
Clarion	4,466,559	12	497	2.69	111.27	126,434	28,307	121,202	27,135
Elk	991,669	2	131	2.02	132.10	27,328	27,558	20,886	21,061
Fayette	33,802,770	93	3,149	2.75	93.16	770,729	22,801	851,378	25,187
Greene	22,432,677	67	1,993	2.99	88.84	539,205	24,037	572,353	25,514
Indiana	12,234,338	44	1,540	3.60	125.88	415,844	33,990	425,224	34,757
Jefferson	6,739,368	23	737	3.41	109 36	235,591	34,957	234,603	34,811
Lawrence	519,251	1	34	1.93	65.48	14,801	28,505	11,264	21,693
Mercer	485,107	2	64	4.12	131.93	24,262	50,014	24,574	50,657
Washington	22,200,558	83	2,515	3.74	113.29	772,381	34,791	774,063	34,867
Westmoreland	39,465,605	111	3,515	2.81	89.06	920,051	23,313	1,003,913	25,438
Venango	30,677	0	1	....	32.60	262	8,541	72	2,347

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

TABLE 3.—*General Cause of Accidents in Pittsburgh, Sewickley and Redstone Seams in Insured Mines in Pennsylvania*, 1925 *to* 1928, *Inclusive*

Total Tonnage 114,973,433 Net Tons

General Cause (1)	Compensated Accidents[a]		Accident Rate per Million Tons		Total Cost[b] (6)	Cost per Million Tons (7)	Total Days Lost (8)	Days Lost per Million Tons (9)
	Deaths and Permanent Totals (2)	All Others (3)	Deaths and Permanent Totals (4)	All Others (5)				
All causes	351	10,896	3.05	94.77	$3,012,728	$26,204	3,192,774	27,770
Mining machines	4	447	0.035	3.89	110,857	964	84,830	738
Mine cars and motors	72	2,738	0.626	23.81	686,841	5,974	681,185	5,925
Electricity	10	68	0.087	0.59	19,800	172	63,456	552
Explosives	6	80	0.052	0.70	46,302	403	60,664	528
Gas and fires	3	19	0.026	0.16	9,044	79	18,757	163
Falls of roof and coal	231	3,859	2.009	33.56	1,597,622	13,896	1,837,700	15,984
Handling materials	6	1,295	0.052	11.26	128,862	1,121	91,790	798
Hand tools	2	1,201	0.018	10.45	183,238	1,594	154,094	1,340
All other causes	17	1,189	0.148	10.35	230,162	2,001	200,298	1,742

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

TABLE 4.—*General Cause of Accidents in Upper, Lower and Double Freeport Seams in Insured Mines in Pennsylvania,* 1925 *to* 1928, *Inclusive*

Total Tonnage 62,680,005 Net Tons

General Cause	Compensated Accidents[a]		Accident Rates per Million Tons		Total Cost[b]	Cost per Million Tons	Total Days Lost	Days Lost per Million Tons
	Deaths and Permanent Totals	All Others	Deaths and Permanent Totals	All Others				
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
All causes	213	6,395	3.40	102.03	$1,983,532	$31,645	2,016,236	32,167
Mining machines	5	400	0.080	6.38	114,929	1,834	93,357	1,489
Mine cars and motors	49	2,321	0.782	37.03	535,207	8,539	537,956	8,583
Electricity	5	52	0.080	0.83	24,418	390	31,179	497
Explosives	13	93	0.207	1.49	109,242	1,743	114,163	1,821
Gas and fires	6	19	0.096	0.30	24,813	396	37,828	604
Falls of roof and coal	118	1,733	1.883	27.65	868,052	13,849	937,223	14,952
Handling materials	3	689	0.048	10.99	84,232	1,344	51,529	822
Hand tools	1	496	0.016	7.91	87,740	1,400	71,386	1,139
All other causes	13	592	0.208	9.45	134,899	2,150	141,615	2,260

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

TABLE 5.—*General Cause of Accidents in Upper, Middle and Lower Kittanning Seams in Insured Mines in Pennsylvania*, 1925 *to* 1928, *Inclusive*

Total Tonnage 90,704,290 Net Tons

General Cause	Compensated Accidents[a]		Accident Rates per Million Tons		Total Cost[b]	Cost per Million Tons	Total Days Lost	Days Lost per Million Tons
	Deaths and Permanent Totals	All Others	Deaths and Permanent Totals	All Others				
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
All causes............	297	10,365	3.27	114.27	$2,687,358	$29,628	2,757,501	30,401
Mining machines.........	3	487	0.033	5.37	112,997	1,246	82,934	914
Mine cars and motors.....	68	3,906	0.750	43.06	762,773	8,409	717,465	7,910
Electricity..............	7	81	0.077	0.89	18,652	206	47,711	526
Explosives..............	24	193	0.265	2.13	182,723	2,015	222,027	2,448
Gas and fires............	0	15	.....	0.17	1,387	15	564	6
Falls of roof and coal.....	184	2,921	2.028	32.20	1,242,565	13,699	1,406,626	15,508
Handling materials.......	1	1,071	0.011	11.81	82,915	914	50,569	558
Hand tools..............	3	790	0.033	8.71	129,642	1,429	109,457	1,207
All other causes.........	7	901	0.066	9.93	153,704	1,695	120,148	1,324

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

Table 6.—*General Cause of Accidents in Clarion, Brookville and All Other Seams in Insured Mines in Pennsylvania, 1925 to 1928, Inclusive*

Total Tonnage 4,759,364 Net Tons

General Cause	Compensated Accidents[a]		Accident Rates per Million Tons		Total Cost[b]	Cost per Million Tons	Total Days Lost	Days Lost per Million Tons
	Deaths and Permanent Totals	All Others	Deaths and Permanent Totals	All Others				
(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)
All causes..............	16	657	3.36	138.04	$206,947	$43,482	179,413	37,697
Mining machines.........	..	25	.....	5.25	4,976	1,045	2,592	544
Mine cars and motors.....	3	203	0.630	42.65	53,666	11,277	41,662	8,754
Electricity..............	1	6	0.210	1.26	5,641	1,185	6,124	1,287
Explosives..............	1	17	0.210	3.57	7,493	1,574	10,710	2,250
Gas and fires............	..	3	.....	0.63	440	92	195	41
Falls of roof and coal.....	9	168	1.891	35.30	95,952	20,160	84,861	17,830
Handling materials.......	..	72	.....	15.13	6,247	1,313	3,440	723
Hand tools..............	..	70	.....	14.71	9,171	1,927	7,718	1,622
All other causes..........	2	93	0.420	19.54	23,361	4,909	22,111	4,646

[a] Waiting period of 10 days in 1925–1927 and 7 days in 1928.

[b] Includes funeral cost and all medical costs.

in a hoisting shaft and which could have no connection with the coal itself. The accident rate for catastrophes is shown as follows:

	Tons of Coal, 1916–1930	Catastrophe Fatals	Catastrophe Fatality Rate per Million Tons
Low-volatile	535,494,800	95	0.177
High-volatile	1,631,410,668	461	0.282

It is noted at once that the rate per million tons of coal for the low-volatile district is only 62.7 per cent as high as is the rate for high-volatile, and is the only place at which the fatality rate is materially different in the two districts. In looking for a reason for this, we find that coal dust was a factor in most of the explosions. The inference, therefore, is that the high-volatile coals are more susceptible to explosions of dust. With the exception of the catastrophe rate, there is little difference in the accident frequency between the low-volatile and high-volatile districts.

EXPERIENCE COMPILED BY GROUPS OF ACCIDENT CAUSES

It was suggested that this same experience compiled by groups of accident causes might help to solve the question asked, therefore Tables 3 to 6 were prepared, showing this information by general causes of accidents and by groups of seams of coal.

The two prolific general causes of mine accidents are haulage and falls of roof and coal. It is interesting to compare the accident rates of these two general causes in the different groups of coals. Haulage, involving mine cars and motors, is responsible for 0.626 deaths and 23.81 all others per million tons in the Pittsburgh seams and 0.782 deaths and 37.03 all others in the Freeport seams, while the Kittanning seams have 0.750 deaths and 43.06 all others and the other seams show 0.630 and 42.56. The number of days lost per million tons shown in the last column on these tables is just as erratic. Pittsburgh shows 5925, Freeport 8583, Kittanning 7910 and Clarion 8754.

The corresponding figures for falls of roof and coal are hard to explain. The Pittsburgh seams have a death rate per million tons of 2.009 and for all others 33.56. The Freeport seams have a death rate of 1.883 and 27.65 for all others. The Kittanning seams have a death rate of 2.028 and 32.20 for all others, and the Clarion seams have a rate for death of 1.891 and 35.30 for all others. Days lost per million tons for these groups of seams for this same cause are Pittsburgh, 15,984; Freeport, 14,952; Kittanning, 15,508 and Clarion, 17,830. Why the Pittsburgh seams, with all of their draw slates, have a lower death rate from this cause than the Kittanning seams is hard to explain. A careful study of the details in these tables raises many similar questions just as hard to answer.

Many opinions have been expressed as to why some of these things are so, but the reasons given are nearly as numerous as are the individuals who vouchsafe their ideas. One reason assigned as having an influence on the accident frequency is the height of the seam of coal. That this theory was pretty generally believed is evidenced by the fact that soon after schedule rating of coal mines for compensation insurance rates was started in Pennsylvania, an item was introduced in the schedule for height of seam, which carried a graduated charge for heights of coal from 35 to 101 in. and over.

TABLE 7.—*Fatality Rate According to Height of Seam*

Per Million Tons Mined

Height of Seam, In.	1916–1919	1920–1923	1916–1923
33	2.18	2.50	2.36
36	2.09	2.28	2.19
39	2.53	1.93	2.21
42	2.32	2.71	2.50
45	3.05	2.00	2.51
48	2.20	1.90	2.07
51	2.07	2.50	2.25
54	1.38	2.02	1.67
57	2.80	2.60	2.70
60	2.75	2.27	2.53
63	2.70	1.80	2.32
66	2.35	2.42	2.37
69	2.55	1.72	2.25
72	2.56	3.05	2.78
75	2.50	2.50	2.50
78	2.20	1.81	2.00
81	2.70	2.00	2.35
84	3.15	2.15	2.71
87	2.13	3.20	2.50
90	2.45	1.93	2.24
96	2.28	2.27	2 28
102	1.48	1.80	1.64

Soon after the introduction of this item in the schedule, a study was started on the fatality rate per million tons of coal by heights of seam. This study was continued until we had eight years' experience, from 1916 to 1923. The first curve plotted showed the years 1916 to 1919; the second showed the years 1920 to 1923; the third showed the years 1916 to 1923. The results of this study are shown in Table 7. The inside fatality rates per million tons of coal mined are given at intervals of 3 in. The fatalities occurring outside the mine were not used, because the height of seam could have no influence upon them.

Plot these curves and note the pronounced saw-tooth effect and probably you will come to the same conclusion arrived at by a committee on schedule rating; namely, that there is no relation between the height of coal and the accident frequency.

Geological conditions of the different seams of coal and the same seam of coal in different sections of the state have been assigned as having a relation to the accident frequency, and we all recognize this difference in hazards in different parts of the state, but we also know that mines operating in localities where most hazards exist have some of the lowest accident rates on record. Pitch of seams or grades have also been assigned as a cause for higher accident rates, particularly as reflected in haulage. There is no doubt that this is true in many places, yet we know mines that have most menacing grades and have very low accident rates from this cause.

Control of Accidents an Operating Problem

It will probably be taken for granted, from what has been said in this paper, that the writer believes that the volatile matter in our bituminous coals has no relation to the accident frequency and severity. This is so, and neither does he believe that the height of seam, the geological conditions, the pitch of seam with its corresponding grades, or any other special condition, necessarily has its relation to the frequency of accidents. The writer rather adheres to the idea that the whole question of accidents and accident frequency is an operating problem and never will be solved until it is considered in that light. Regardless of the hazardous conditions in mining, be they so-called natural or brought about by man, the coal operator, a most resourceful person, can meet these conditions and operate his mine safely. The operator who does meet them, and there are those who do, demonstrates that the mining of coal is not necessarily a hazardous occupation.

DISCUSSION

(*John J. Lincoln presiding*)

[This discussion refers also to the paper by R. F. Roth, beginning on page 20.]

R. M. Lambie, Charleston, W. Va.—It is often stated that there are more accidents in the low-volatile fields than there are in the high-volatile fields per thousand men employed and million tons produced. Our records, of course, prove this to be correct, and it is due, in my opinion, to the fact that there are more and greater hazards in the low-volatile fields of our state than exist in the high-volatile fields. It certainly is not due to the fact that the high-volatile officials are more competent, or that the miners in the low-volatile fields are not as competent as in the high-volatile districts.

One reason for a higher accident rate in the low-volatile fields is that the coal extraction is higher than in many other fields. Many states and districts do not have any pillar extraction, and anyone familiar with mining knows that the hazard is tremendously increased by the extraction of pillars.

Reports from various states make no mention of the fact that they have only solid mining, therefore any comparsion on a man-hour exposure, or million tons produced, is not fair to the sections that endeavor to obtain complete extraction.

The records of the West Virginia smokeless fields continue to show improvement, regardless of the fact that 40 per cent of the production is being obtained from final robbing under extremely dangerous conditions. Several splendid records have been made in the smokeless fields by the United States Coal & Coke Co., The Koppers Coal Co. and the Consolidation Coal Co., as well as several others.

It is doubtful whether all states list accidents uniformly, therefore I would suggest that some method be established whereby every state would list all accidents.

E. Ramsay, Birmingham, Ala.—In a few words I want to tell a little of what we are doing in Alabama in the way of reducing accidents. The records given here today of what is being done in West Virginia demonstrate conclusively that good results can be accomplished in the way of reducing mine accidents by hard work.

In Alabama we have as chief mine inspector, W. B. Hillhouse, a hard and forceful worker, with whom the coal operators of Alabama, the U. S. Bureau of Mines and the Joseph A. Holmes Safety Assn. work whole-heartedly.

Last year Inspector Hillhouse and the Alabama Mining Institute were awarded Certificates of Honor by the Joseph A. Holmes Safety Assn., Washington, D. C., for their accomplishments in the marked reduction of accidents in Alabama mines. These certificates were much deserved by both the Institute and Mr. Hillhouse and I am informed that never before in the history of the United States have such certificates been awarded to a state mine inspection department or to a mining institute.

Being chairman of the Mine Casualty and Mining Institute Committee of the Alabama Mining Institute, I was informed before leaving home to attend this Bluefield meeting that Alabama has had only 11 fatalities this year to date, and if this record continues for the remainder of the year, and based on the 11 deaths we have had up to the present, Alabama will rank high among the leaders in low number of fatalities in coal mines of the United States. These figures show the remarkable work now being done in Alabama towards reducing mine accidents, and I feel safe in saying that Alabama will produce half a million tons of coal per fatality for the year 1931.

T. G. Fear, Fairmont, W. Va.—In considering the frequency of mine accidents you must also consider the severity. One means nothing without the other.

We are mining in four states and ten seams, and we find no difference between high and low-volatile mines. In the low-volatile mines of the Pocahontas field the frequency was 36 as compared with 62 in the Pittsburgh and northern West Virginia fields. The severity was 18 in the Pocahontas field as compared with 19 in the Pittsburgh and northern West Virginia field.

An important matter is the cost per ton for compensation which for the first eight months was 1.6¢ against 2.1¢ per ton in the Pittsburgh and northern West Virginia field.

The miners in our mines are cautioned to be careful, and if you visit one of the Consolidation mines you will find the men telling you to be careful. They tell each other to be careful. It costs nothing and it is one of the greatest aids towards safety.

C. E. Lawall, Morgantown, W. Va.—Has your study brought out any comparative data as to the number of accidents occurring during the good and bad years in the coal industry? Some operators say that the greater the coal production, the greater is the number of accidents, while other operators say that the reverse of this is true. What are the facts of the matter?

R. F. Roth.—In making up my paper I anticipated such a question and thought that the best way to answer it would be through the eye by means of a graph. This graph (Fig. 14) shows the total number of accidents, both fatal and nonfatal, in West Virginia from 1921 to 1930, the broken line representing the high-volatile district and the solid line representing the low-volatile district; it also shows the production in millions of tons.

In 1921 across 1922 there is a pretty close relation between tonnage and accidents, both downward; then slightly upward. Then from 1924 the high volatile is quite even across to 1930, and the variation of accidents to tonnage performance right on through the several years is evident. The lines in general are by no means parallel and it can readily be seen that there is no fixed relation or ratio between the two elements. In some cases reduced tonnages have corresponding high accident rates and in other instances the years of increased production show a decrease in accidents.

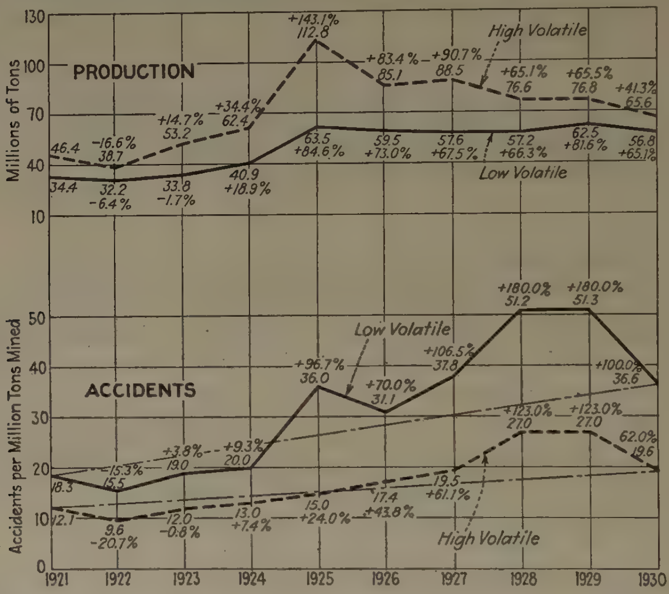

Fig. 14.—Relation between tonnage and number of accidents.

The percentage relation for each year as compared with the year 1921 is marked on the curves. The figures also show conclusively that there is no fixed relation between tonnage and accidents.

T. G. Fear.—My observation about the mines in Somerset partly answers the question, and that is the matter of exposure. Then comes the question of machinery, which means that the men are producing more tons per man, and that has something to do with the accident cost per million tons, making it lower. I think the accident cost is largely one of exposure.

G. S. Rice, Washington, D. C.—If we want to get the causes of the accidents in mines we will have to gather the statistics more in detail and place them on a basis of exposure by groups, such as timbermen and loaders, which means getting the number of men exposed to the particular hazard, and then rate the accidents which have occurred under those different classes.

By grouping all underground accidents together, you do not have an opportunity to find out how and where the accidents occur. I cannot agree that it is wise to rate accidents on the basis of tonnage produced. When this is done, the factors of mechanization and efficiency are introduced, which obscure the facts of relative exposure of men to the hazard. This paper, while containing interesting suggestions, seems unsatisfactory in respect to attempting to compare accident rates on the basis of the volatile content in the coal of different districts. Apart from the matter referred to above, basing the accident rates on coal production, it is doubtful whether it is wise to base accidents on some mysterious effect of chemical composition or origin of the coal, when there are what seem to be more pertinent physical factors of condition of roof, thickness of coal bed, dip of bed, explosibility of its coal dust, whether gassy or not, etc. In the Rocky Mountain region I have seen both exceptionally strong and exceptionally weak roof in mines producing subbituminous coals and also the same extremes in low-volatile coals. Further, some of the subbituminous beds dip steeply, and there are similar steep dips and variable conditions in the anthracite districts. So it appears that any difference in volatile ratio of the coals mined has nothing directly to do with the relative hazards in mining of the different fields. Is it not the physical conditions, not the chemical ones, that affect the mining hazards?

D. L. **Boyle**, Johnstown, Pa.—I would like to emphasize Mr. Hosler's closing remarks (p. 42). From my own experience of many years, I know that this summary as prepared by Mr. Hosler is absolutely true and the management must take the stand that accidents must be prevented. If they do this, I am sure that our accident prevention work will be more successful.

G. H. **Deike**, Pittsburgh, Pa. (written discussion).—It is apparent that Mr. Hosler, with records covering mine accident statistics over a four-year period, has been able to present in this very forceful manner facts which will enable the mining men to cease groping for apparent and supposed causes of mine accidents and solve the question of accident prevention in coal mining with definite information which will be helpful in getting the results for which we are all striving.

First of all we recognize the fact that the statistics given, while they do not cover all the bituminous mines in this country nor even in the State of Pennsylvania, yet cover an exposure of 275,000,000 tons, or over 50 per cent of the bituminous coal production in Pennsylvania, will be accepted as indicative of the results in the industry as a whole.

Mr. Hosler has very definitely brought us face to face with the facts which have to be considered in accident prevention and mine safety work. He has told us to discard the alibis and excuses behind which we have taken refuge in the past; namely, that accident frequency and severity were governed by the characteristics and geological conditions of the coal seam, height of seam, pitch of seam, grades—the natural characteristics of the seam of coal worked. It is very gratifying to have him state that from his experience in many years of coal mine insurance rating all the figures and statistics point conclusively to the fact that if the operator will recognize fully the particular conditions at his own operations, he is fully capable of meeting them adequately and operating his mines safely.

I agree with him that by getting all the facts in connection with the conditions to be met and applying the accident prevention methods and equipment which have now become standard practice, combined with proper discipline in having all employees comply with the safety rules and regulations as they are now formulated, we can demonstrate that coal mining is not necessarily a hazardous occupation.

R. N. **Hosler**.—Out of 33,956 accidents there were 817 deaths and 150 permanent totals. We sometimes lose sight of these permanent totals, yet they cost more than an actual death.

Let us take 997 major permanents; 616 minor permanents and 31,369 temporary accidents compensated for, and 40,000 additional accidents for which there was no compensation, but for which there were medical costs. The important thing about these figures is this: here is 31,000 in one class and 40,000 in another, nearly all of them potential deaths, and in the aggregate those 31,000 compensable accidents cost more than the 817 deaths.

The way to attack the question of safety is to cut down accident frequency, because every time an accident occurs it may mean anything from a death to a scratch that merely requires some medical dressing.

Coal Evaluation and Preparation

By Thomas F. Downing, Jr.,* Philadelphia, Pa.

(Bluefield Meeting, October, 1931)

When examining a coal property it is customary for the engineer to take channel samples at several coal faces. In doing so the extraneous matter, or partings, which can be removed by hand, or mechanically, with the facilities at hand, or proposed, is rejected. Where there is doubt concerning a stratum it is usually rejected from the main sample but part is taken for separate analysis. The samples are forwarded to a chemist for proximate analyses and determination of British thermal units, and, sometimes, ash fusion.

Consideration is given all available commercial analyses but frequently the owner presents only the favorable ones. From the results thus obtained a general evaluation is made, monies are loaned or expended, markets are determined, and realization is estimated. In many instances, such grades are not substantiated and it is time we looked deeper for a basis which will prove more practical.

Institute committees are working hard on analytic and use classifications and much valuable information has already been assembled and distributed. However, no single grade can be applied to any particular bed or region. Much study has been given the Pittsburgh No. 8 seam, yet that bed is subject to material differences in a single section; *i. e.*, some strata of a face may differ from others in structure and analysis, particularly in sulfur content. From some localities it yields a good carbonization fuel, while in others it is chemically unfit for such use.

Traveling south to the Logan, W. Va., field we find the Cedar Grove (Island Creek) seam a hard, gray splint coal. In the same field, and not far distant, the same bed changes, both structurally and chemically, and consists of excellent gas coal. In Illinois the moisture content of a bed may vary greatly in different localities. We have seen mines in which part of the coal is excellent for special purpose use but is spoiled by the product from another part of the same mine.

Many other illustrations could be given but these are sufficient to show that application of any grading of coal must be distinctly local. The actual designation of grade can be given only after detail study of the component parts of the seam, or seams, on the property under consideration. The grade given can be substantiated in practice only by choosing mining methods and preparation facilities fitting the results of such a study.

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* Consulting Mining Engineer.

EFFECT OF MINING METHODS ON GRADE OF COAL

In most bituminous mines the first step in mining is the machine cutting of the coal. Machines are available for cutting in any part of the seam and the cutting point is usually decided by the physical characteristics of the bed or the location of impurities. The writer has never found a case where the chemical qualities of the coal were given any consideration in the choice of cutting position, yet improper cutting can entirely change the grade given by the reporting engineer. To clarify and impress the importance of this statement three actual instances are here given.

An operator mined a seam that was about 6 ft. thick and contained a middle slate parting which ran about 1 to 4 in. in thickness. The bottom coal was softest so he cut on the bottom and shot the whole bed at once. His tipple preparation facilities consisted of screens and picking tables. Analyses of channel samples showed an ash content of 5.4 per cent. and the 1¼-in. slack ran consistently under 7 per cent. ash and was acceptable, and sold, for carbonization use.

As more of his development entered the area of thicker parting there was a tendency toward increased ash in the slack size. He decided to purchase arc wall machines and cut just above the parting. The top bench was shot and loaded first. When the bottom bench was shot the parting broke into large slabs which were easily removed and gobbed in the rooms. From an operating standpoint the change was a big success. More gob was left in the rooms and the picking table force was materially reduced.

Presently complaints began drifting in from consumers, which were not treated seriously at first. The water-gas plant said that the coal was more friable. The operator said it was the same coal that he had always shipped. The ash in the slack increased. The operator claimed that the coal was cleaner than it had ever been. Run of mine ash fluctuated between 3 and 12 per cent. "Why, there is not half that much ash in the whole seam, and I could not get it if I shipped parting and all," said the operator. However, complaints continued, he lost some of his best business, and some shipments were refused. When steam plants began to complain, the perplexed owner called in help.

Analyses were closely studied and examinations made of the faces. There was no question but that the change must be in the seam, so every stratum was sampled and analyzed separately. It was found that most of the top bench contained 7.8 per cent. ash while one band ran as high as 9.3 per cent. One foot of the bottom bench contained 4.2 per cent.

ash while the remaining 2 ft., in which he had previously cut, ran as low as 2.5 per cent. When he cut on the bottom the whole seam was proportionately mixed in shooting and loading but when the benches were loaded separately half of the product was high in ash while the ash of the remainder ran under the average of the seam. Needless to say, the bottom cutters were put back into the mine.

This illustration is cited in considerable detail because improper cutting entirely changed the grade of coal, produced an erratic ash, lowered realization and yielded a product which the finest mechanical cleaning plant could not make as consistent as it was under the old system of mining.

The seam at another mine had two "rash" partings. One was cut out by an arc wall machine which could not reach the second one. Relatively inexpensive changes were made in the cutter head and the machine range was changed so that both bands of impurities were cut out and discarded. As a result the ash content of the slack coal fell from 11 to 6 per cent.

In another case the bed carried a foot of top coal which was high in ash and sulfur. The coal was cut on the bottom and it was impossible to keep the impure coal from mixing with the other in shooting. Top cutting machines were purchased and the coal cut under the bad stratum. After the cut was loaded, the top was taken down and loaded out from the entries or gobbed in the rooms. The product was cleaner, considerable expense in tipple cleaning and waste disposal was saved and the efficiency of the loaders was increased.

Roof control, proper relation of room-and-pillar widths to overburden, good pillar lines, and correct shooting all have a direct bearing on the grade of the product and bring larger returns by producing cleaner coal and increased percentages of prepared sizes. These subjects are too lengthy for discussion here; they are generally understood by mining men, though they are overlooked by some.

Intelligent study of coal beds and surrounding conditions and practical application of correct mining methods would solve many operating problems and often would give an acceptable product without large capital charges in mechanical preparation.

Mechanical Preparation

Some seams require mechanical preparation to make them acceptable to the trade. In such instances too much study cannot be given the material to be treated. Often plants are installed after incomplete tests and much more of the product is prepared than necessary. Practically no attention is given the chemical contents of the strata. Stress is rightly given the percentage of impurities in the size, or sizes, to be treated but more detail as to the physical conditions of the extraneous material is necessary.

Suppose it is proposed to erect a washery to treat the 2-in. minus size from a seam having two or more partings. Gravity tests should be made of samples taken from the face of each stratum to be discarded, as well as of any roof or floor which may get into the raw product. These should be compared with similar findings in regard to the pure coal. Attention should be given the fracture and shape of these impurities and thus some idea can be gained of the sizes in which they are likely to be found most abundantly.

Tipple samples should be taken at intervals daily for at least a week and a properly quartered part of each day's sample divided for tests. The samples should be hand-screened into at least six sizes. The sizes above ½ in. can be hand-picked. All others should be given float-and-sink tests. If results of the several days are not fairly comparable, the reasons should be sought. If double shooting or middle cutting is done it may be necessary to divide the day into two sample periods in order to get the maximum amount of impurities to be removed at any one time. The range between maximum and minimum duty should also be a big factor for consideration in choice of equipment.

The results of such tests are sometimes surprising. Almost always a considerable part of the product needs no preparation. Sometimes the intermediate sizes are relatively clean, particularly so when two strata have different structures. Sizes which do not require preparation should not be treated, but should be by-passed. A little more screening would often save considerable capital expenditure and operating charges in the preparation plant.

Considering every angle, it is conservative to say that the capital invested in preparation plants in this country is far beyond that required to accomplish the results that are realized. The by-passing of the comparatively clean coal would also greatly neutralize the moisture in the washed coal when the sizes are reassembled.

The heavy pitching anthracite beds present different operating problems but detail study of the seams is just as important because several beds differing greatly in physical characteristics are frequently prepared in common units in the same breaker. In a written discussion on classification last year[1] the writer held that where the strata of a coal bed differ in structure they differ analytically too. The first illustration in this paper also stresses that fact.

In anthracite one can hardly imagine a greater physical difference than exists between the Buck Mountain and the top bench of the Mammoth seams. It is almost as great as that between the gray splint coal and the Sewall seam of West Virginia. The three splits of the Mammoth bed itself differ greatly in structure and value. The extraneous material in the many seams also varies greatly. The "bony" coal in one seam may be near the specific gravity of the coal of another bed while there is a slate parting in the Mammoth seam of which the gravity closely approximates that of the coal of the Holmes bed in some localities. There is a strong possibility that some of the seams should not be mixed with others in a common preparation unit and that proper segregation in washing might result in better preparation and less loss of coal in the reject.

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[1] T. F. Downing, Jr.: Coal Classification. *Trans.* A. I. M. E., Coal Div. (1930) 714.

ASH FUSIBILITY

Advanced practices in combustion have brought to the front consideration of the fusibility of coal ash and thereby have materially influenced use classifications. As the melting temperature of the ash in coke is the same as in that of the coal from which it is derived, coke is also a contributing factor in class whether it be used for gas manufacture or as a domestic fuel.

Pyrites are naturally high in iron content and fuels containing them are therefore subject to low melting temperatures. However, coals free of pyrites are also in the low fusion class.

The amount of ash is not a large factor but the fluxing constituents of the ash are. Pure coals with high fusibility ratings are sometimes found otherwise in practice. This is due no doubt to the low fusibility of the extraneous material which mixes with the coal in mining. In many instances the examining engineer should take samples of all partings and any extraneous ash that may become mixed with the commercial product and have them subjected to fusibility tests.

To elaborate this thought, let us consider anthracite coal which, as a class, has the highest and most consistent softening temperature of all the solid fuels. Practically all anthracite coal is washed and may be considered free of pyrite in shipment. The smaller sizes are hardest to clean and contain higher percentages of extraneous material as the size decreases.

The ash in the pure coal fuses at temperatures of 2900° F. and upward. Yet those who have used this fuel have found that it clinkers in small domestic plants whose firebox temperatures are under that point; and the smaller the size used, the more clinker is found. Whence come those fluxing constituents in the ash? From the bone of some particular seam or from partings, top, or bottom? Or are they a result of mixing the impurities of one bed with those from another? It would take the combined efforts of an experienced engineer and a competent chemist to find out, but it would certainly seem important to the anthracite industry to know. Knowledge of the facts and remedial methods, if possible, would certainly make their product more acceptable to the trade, particularly now when the tendency is toward increased use of the smaller sizes.

Considerable study has been given to the effect of ash of low fusibility but little practical application has been given to control possible limits in shipments of coal. A. C. Fieldner[2] gives a table showing the difference in softening temperature of ash between raw and treated coals. The coals were floated in the laboratory. Of the 18 tests shown, washing lowered the fusion point in 4. The other 14 show increases varying from 20 to 440°. In one test of the Miller bed, washing would increase the softening temperature from 2470 to over 2910°, while another test of material from the same bed, from the same county, would decrease it from 2930 to 2620°. From the results shown in the table it would appear that mechanical preparation, from an ash fusion standpoint, may raise or lower the grade of coal. It would have little effect on some coals, some would be improved, while others should not be treated at all.

Such information emphasizes the main point the writer is trying to make in this paper; *viz.*, the importance of more intelligent detailed study of the product at the face.

Summary

Just what part of the bed, or beds, causes the greatest amount of ash trouble? And which, if any, have relatively low-temperature ash-fusing points? Can they be controlled without expensive mechanical preparation by changing mining methods and practices? And if not, will such preparation increase or decrease the value of the product?

All of these questions must be intelligently answered before a definite assignment to a grade can be applied to any particular coal, and the studies must be made from face samples with due consideration given to chemical content and mining methods.

DISCUSSION

(*Howard N. Eavenson presiding*)

C. Scholz, Charleston, W. Va. (written discussion).—Mr. Downing says that he "has never found a case where the chemical qualities of the coal were given any consideration in the choice of cutting position." I can cite at least ten mines where such consideration is given.

I developed a mine in the Eagle seam in 1920 where the seam had a section as in Table 1, using undercutting machines, but as the workings had advanced some 500 ft. under cover the seam thinned down and complaints were received about the high ash content. Investigation was made by sampling the three benches separately, not only in the face but at the mouth of the mine as well. The average face sample showed an ash content of 4.6 per cent. The result of this investigation was the adoption of overcutting machines. The top bench was wasted and a second cut was made below the bone, with the result that a most excellent quality of coal was produced, which

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[2] A. C. Fieldner: Influence of Washing Coal on Coke Properties and on Gas and By-product Yields. U. S. Bur. Mines *Rept. of Investigations* 3020 (1930) 10.

Table 1.—*First and Revised Mining in Eagle Seam Mine*

Stratum	First Mining (Total Depth, 6 ft. 4½ in.)			Stratum	Revised Mining (Total Depth, 4 ft. 10½ in.)		
	Depth	Position of Cut	Ash, Per Cent		Depth	Position of Cut	Ash, Per Cent
Coal	34″		8	Coal	12″	—	15
Bone	2½″			Bone	2½″		
Soft coal	28″		2.7	Soft coal	24″	—	2.7
Hard coal	24″	—	3.5	Hard coal	20″		3.65
Average ash			4.6				5.8

met the most critical test of the by-product customers. This information was passed on to another company working in the same seam, which abandoned undercutters and used overcutting machines with most satisfactory results.

R. P. Hudson, Wayland, Ky.—The author of the paper has dealt mainly with the face preparation of coal and has illustrated the fact that, in many cases, the chemical composition of the shipped coal can be controlled without the erection of expensive mechanical preparation plants. There is a paucity of data on the subject of face preparation of coal and Mr. Downing has made a distinct contribution to the literature of the subject. It has been shown that the mining methods employed may have a material effect upon the composition of the coal produced. Unless the operator has accurate knowledge of the composition of the various sections of the seam being mined he cannot control the quality of the coal arriving at the surface, and, in some cases, he cannot control the quality of the shipped coal.

Mr. Downing shows that true face preparation is an impossibility without a scientific study of the coal bed. Face preparation cannot be achieved by simply disciplining the miner at the working face; it can only be realized by adopting standardized methods and practices based upon carefully tabulated data comprising sectional studies of the composition of the coal seam. The success of face preparation does not depend primarily upon the miner; it depends upon the operator and comes through the application of scientific principles, experiment and intelligent inquiry.

It is true, as the author says, that some seams require mechanical preparation in order to make the product acceptable to the trade. Face preparation is not applicable in all mines. However, it is equally true that mechanical preparation plants are often erected without sufficient advance study of how much mechanical preparation will accomplish. This is also true of the erection of picking tables.

The author mentions a fact that many have observed in practice—that there is often a wide difference in the fusibility of the ash of face samples and the ash fusibility of the shipped coal. This is due, of course, to the fact that the shipped coal contains extraneous material with a divergent ash-fusibility value. I would suggest, however, that in the evaluation of coal little or no attention should be given to the fusibility of the ash. Although it is well to know the temperature at which the ash fuses, as determined by laboratory tests, it should be borne in mind that this temperature may not have any practical or direct relationship to the temperature at which the ash clinkers in the fire box of a domestic or industrial furnace. In laboratory tests we operate upon a sample consisting of a uniform mixture of the ash, but almost never is there a corresponding uniform mixture of the ash in the fire box of a furnace. Therefore we cannot correlate the temperature at which the ash fuses in laboratory tests with the temperature of clinkering in the actual utilization of the fuel. Laboratory fusibility tests have their place and value in fuel technology and every operator should know the fusibility of the ash in the coal seam and in the shipped coal, and he should know also whether or not surface cleaning will increase or decrease the fusing temperature of the ash.

The same information should be at hand in regard to sulfur. When the sulfur in a seam of coal is practically all organic sulfur, it is obvious that surface cleaning will increase the total sulfur content of the shipped coal, while simultaneously the fusing temperature of the ash may be increased. In a case of this kind it must be decided which is the more valuable product, a comparatively high-ash and low-sulfur coal or a high-sulfur coal containing ash with a high fusibility value.

Growth of Coal Preparation in the Smokeless Fields of West Virginia

By T. W. Guy,* Charleston, W. Va.

(Bluefield Meeting, October, 1931)

During recent years, tremendous strides have been made in the economical use of coal. This has resulted in, and to some extent has been a result of, making the fuel specifications more and more rigid. For this reason, all producers have found it necessary to improve their preparation materially, both as to sizing and the elimination of impurities, either by more effective preparation underground or at the tipple and cleaning plant, or both. This cause and its effect in all producing fields seem likely to continue for several years at least.

The splendid inherent qualities of the Pocahontas and New River coals are known all over the world, but their good reputation is only partly due to inherent qualities, and to natural conditions which generally favor loading coal comparatively free from impurities; it is due in large measure to the care and skill used, either with or without the aid of mechanical devices, to ship products consistently low in ash.

The first coal from the West Virginia smokeless fields was mined and shipped near Quinnimont, on New River, in 1871, while the first shipments from the Pocahontas field were in 1883. The development of both fields progressed rapidly. At first activities followed the main lines of the Chesapeake & Ohio and Norfolk & Western railroads, and were extended later as branch lines were built from these roads. The Virginian R. R., about 1907, opened up the Winding Gulf territory, which has since become a large producer of smokeless coals from the Pocahontas and the New River measures.

Early Cleaning Methods

In the early days, the tipples were simple wooden structures, with bar screens for separating the fines from the lump coal. All cleaning was done by the miners at the face, or by pickers in the railroad cars. Gates were provided in the chutes by which the flow of coal could be stopped when necessary while the pickers raked the surface of the coal in the car and picked out the impurities. A large percentage of the fine coal or slack was used for making coke at the mines in beehive ovens. The coal over the slack screen was called "screened run of mine." Some large lump was shipped as early as 1888, shipments of bar-screened egg began about 1890, and of nut coal about 1900.

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* Consulting Engineer.

Soon after 1900, in response to a growing demand for a better preparation, there appeared apparatus for reducing degradation and to reduce the ash in the fine coal and in the coke; and a little later, shaking screens, picking tables, loading booms, etc., to improve the preparation of the larger sizes.

In 1901, in Fayette County, the first cable retarding conveyor in this country was installed in order to reduce the breakage due to handling coal in monitors.

In 1903, the first washer in the smokeless fields was installed at the mine of the Crane Creek Coal & Coke Co. at McComas, West Va. It consisted of two Christ jigs, and was installed by the Tamaqua Manufacturing Co. of Tamaqua, Pa. During the same year, the Jeffrey company installed a Robinson tub washer at the Pinnacle mine of the Pinnacle Coal & Coke Co. at McComas. This equipment, with a capacity of 400 tons per day, washed the coal from 1½ to 0.

This was soon followed by other installations of the tubs, and of other types of jigs. In the early days of mechanical cleaning, it was confined in this field, to what was called nut and slack; that is, the fines passing bar screens with spacing varying from 1 to 2 in. Later, on account of sludge losses and moisture, the majority of the plants washing their nut and slack screened out and by-passed the −½-in. material, confining the washing operation to material through 2 to 2½ in. and over ½ in. At the beginning of the present year, however, there were still five plants in the field wet-washing coal from 2 or 2½ in. to 0. The product of the early washing plant was used for making low-ash coke, or if there was an excess it was mixed with the lump coal and sold as mine run, as at that time there was very little market for slack.

Modern Methods

Since about 1890, the demand for prepared sizes of the low-volatile or smokeless coals had continued to increase, and preparation of the larger sizes had constantly grown in importance.

In 1908, the Empire Coal & Coke Co., at Landgraff, built a 3-track steel tipple with shaking screens, picking tables, loading booms, and mixing conveyor for reassembling the sizes after picking. This is said to be the first tipple with picking tables in the smokeless fields, and the first installation of loading booms and mixing conveyor in the United States. It was designed and installed by the Jeffrey Manufacturing Co. From this beginning, the installation of shaking screens, picking tables and loading booms in the smokeless fields progressed rapidly to such an extent that shortly after the war there were very few tipples not equipped to make prepared sizes; and very little mine-run coal was shipped which was not separated into sizes for picking and then reassembled.

In the smokeless fields, the early growth of mechanical cleaning, meaning the separation of impurities from coal by mechanical means, was confined to the Pocahontas field, where up to and including 1910 washers had been installed at 14 mines. These produced in that year 14.7 per cent of the entire production from the Pocahontas field. One additional plant was built in 1912, after which there were no new installations until 1922, when a washer was installed at the Itman mine of the Pocahontas Fuel Co. In 1923, mechanical cleaning equipment was installed at three mines in the Pocahontas field, one of which, at the Crane Creek mine of the American Coal Co., was the first commercial installation of pneumatic cleaning equipment east of the Rocky Mountains. In 1924, additional equipment was installed at five plants in the Pocahontas field, and the first plant for pneumatic cleaning in the New River and Winding Gulf field was installed at the Wyco mine of the Gulf Smokeless Coal Co. Since that time, several mines in the two fields have been added each year to the list of those mechanically cleaning at least a part of their output.

PRESENT EXTENT OF MECHANICAL CLEANING

Early in the present year, the plants with mechanical cleaning equipment numbered 52 in the Pocahontas field as outlined in recent injunction, and 21 in the New River and Winding Gulf field, making a total of 73 plants belonging to 54 companies. These 54 companies produced 73 per cent, and the 73 plants produced about 42.5 per cent of the total smokeless output in 1930. Of the 73 plants, 12 used pneumatic or dry equipment only, and 45 wet equipment only; while 16 used both pneumatic and wet equipment.

Of the 73 plants, 18 are mechanically cleaning their egg coal; 17 wet, and 1 dry; 71 are cleaning stove or nut coal, 60 using wet equipment and 11 dry.

At 58 plants of the 73, pea coal is being cleaned, in 48 plants by wet equipment and in 10 by dry.

At 25 plants of the 73, slack is being cleaned; 5 using wet and 20 dry equipment.

Table 1 shows the production of the 14 mines having cleaning plants in 1910, and the relation of the output of these mines to the other mines, and to the total of the Pocahontas field during the years 1910, 1920, 1921, 1929 and 1930. It is striking that the same mines that installed mechanical cleaning equipment from 20 to 28 years ago produced a greater ratio of the tonnage from the field in 1930 than in 1910. Evidently they have been able to hold their own, not only with the mines in operation at the time they installed their cleaning equipment, but with all of the new mines

TABLE 1.—*Output from Plants in Pocahontas Field*[a]

Year	14 Plants Using Washers Installed Prior to 1911		All Other Plants in Pocahontas Field		All Plants in Pocahontas Field	
	Thousand Tons	Per Cent of Total	Thousand Tons	Per Cent of Total	Thousand Tons	Per Cent of Total
1910	2,501	14.7	14,489	85.3	16,990	100.0
1920	2,797	14.4	16,638	85.6	19,435	100.0
1921	3,054	17.9	14,008	82.1	17,062	100.0
Increase or decrease.......	+257	+9.2	−2,630	−15.8	−2,373	−12.2
1929	4,220	14.0	25,894	86.0	30,204	100.0
1930	3,904	15.3	21,571	84.7	25,475	100.0
Decrease.......	−316	−7.5	−4,413	−17.0	−4,729	−15.7

[a] Only mines included in 1930 Statement of Annual Shipments.

which have been opened and developed in the field since. One of the 14 mines was worked out prior to 1929.

It is also interesting to note that in 1920, when prices were at a maximum, and there was a demand for every car of coal that could be loaded, this group of mines produced 14.4 per cent of the production from the Pocahontas field; and in 1921, which was a year of severe depression in general business as well as the coal business, this group of mines produced 17.9 per cent of the production from the Pocahontas field. Stated another way, in the year of depression, the production from this group of mines showed an increase of 9.2 per cent over their own 1920 production, while the remainder of the field showed a decrease of 15.8 per cent. In 1930, this group produced 15.3 per cent of the total Pocahontas production against 14 per cent in 1929.

Table 2 shows a comparison of the output from all mines having cleaning equipment in 1930 with the output of the same mines in 1929. It also shows a similar comparison of the tonnages of the 17 mines that mechanically clean their nut, pea and slack.

The group of mines cleaning all of their small coal showed a smaller decrease in 1930 than the group of mines that mechanically cleaned a smaller percentage of their output. Also, both groups showed some advantage over the plants without any mechanical cleaning, although the advantage was not so great as was shown by the group cleaning in 1920 and 1921.

In comparing 1929 and 1930 with 1920 and 1921, it should be remembered that 1921 and 1930 were both years of depression in general business, which in both instances, began during the preceding year. In the coal business, 1920 represented unusual demand and very high prices

TABLE 2.—*Output of Coal from Pocahontas Field in* 1929 *and* 1930

Year	47 Plants Having Cleaning Equipment in 1930		All Other Plants in Pocahontas Field		All Plants in Pocahontas Field	
	Thousand Tons	Per Cent of Total	Thousand Tons	Per Cent of Total	Thousand Tons	Per Cent of Total
1929	15,858	52.5	14,346	47.5	30,204	100.0
1930	13,576	53.3	11,899	46.7	25,475	100.0
Decrease.......	−2,282	−14.4	−2,447	−17.1	−4,729	−15.7
	17[a] MINES CLEANING NUT, PEA AND SLACK					
1929	5,441	18.0	24,763	82.0	30,204	100.0
1930	5,106	20.0	20,369	80.0	25,475	100.0
Decrease.......	−335	−6.2	−4,394	−17.7	−4,729	−15.7

[a] Only mines included in 1930 Statement of Annual Shipments.

while in 1929 prices were very low but the demand was sufficient to carry the production of the smokeless fields nearly 10 per cent above the best preceding year.

Those who have been more or less disappointed recently with the poor returns from mechanical cleaning may find some encouragement in the record of those Pocahontas mines, which have been cleaning mechanically for over 20 years.

EQUIPMENT IN USE

The following types of equipment have been installed in the Pocahontas field during the last few years:

Pneumatic Equipment:—Pneumatic separators—American Coal Cleaning Corp.; Arms air concentrators—Roberts & Schaefer Co.; Peale-Davis Pneumo-Gravity separator—Pennsylvania Mining Machinery Corporation.

Wet Equipment:—Rheolaveur washer—Koppers-Rheolaveur Co.; Simon-Carves washer—Link-Belt Co.; Pittsburgh jigs—Pittsburgh Coal Washer Co.; Hydroseparators—Roberts & Schaefer Company.

STANDARDIZATION NEEDED

The writer believes that the next steps in bituminous coal preparation will be:

1. A drive for uniform quality in the products shipped. Practically all complaints from consumers specify a lack of uniformity in ash, sizing, fusing or other characteristics. They insist that uniformity is necessary for uniform results, and that uniform results are necessary for economy. The consumer will not remain sold on

any product that varies materially in quality, therefore the producer should concentrate on uniformity and pay less attention to average characteristics. This seems to offer the greatest inducement to the consumer to pay the necessary cost of preparation.

2. The close coordination of preparation and marketing plans. In the mining, preparation and marketing of the output of a mine, the objective, of course, is to produce the largest net profit. Throughout the three operations, each step should be planned to this end. When this is lost sight of, it frequently happens that ash reduction is carried beyond the market requirements in some cases, and not far enough to meet the requirements in others, the result being a heavy loss in both cases. Making a concession on size, or agreeing to meet lower ash specifications, etc., may increase preparation or mining cost out of all proportion to the value of the business involved. Such difficulties may be avoided by carefully determining beforehand what products can be made economically, and what costs would be necessary to meet certain changes in specifications.

3. Cooperation of producers and consumers in establishing standards for the preparation and marketing of coals. The question of establishing standards for coal preparation in the bituminous fields has been discussed frequently. Every consumer of coal, every coal salesman, every mine official and every coal miner has often felt exasperated because there was no answer to the question, "*How* clean is clean coal?" Recently, Mr. Eavenson, Chairman of the Coal Division of the A. I. M. E., requested the American Standards Association to cooperate with the Institute in formulating standards for clean coal.

The idea seems to prevail that the question is too complicated and difficult for solution, because of the wide variation in quality and characteristics, and the wide variation in uses, of bituminous coals. This might be true, but it should be remembered that the products of practically all industries have been more or less standardized, and that such standardization has always been an important factor in stabilizing the markets, in reducing costs and building up good will.

The tremendous sale of goods in standard packages, at prices higher than the cost of similar goods that are without the uniform quality guarantee which the package and trade mark gives, demonstrates the tremendous popular appeal of standardized products. Back of this demand are sound economic principles, the first and most important of which is that uniform products give uniform results in use; another is the confidence and satisfaction that comes from feeling that you know what you are getting for your money.

A prominent mine official in the Pocahontas field has expressed his belief that if definite standards could be adopted in the Pocahontas field, by which a producer could know definitely that he had shipped the product which he sold and by which the consumer could know that he had received what he bought, such standards would be worth several hundred thousand dollars each year to the operators of the field.

In the absence of standards, there is a tendency for each concession made by an individual operator in size or quality of the product to be forced upon his neighbors and competitors, with the natural result that everyone finds himself continually forced to give more and more for the same returns. At the same time the consumer, owing to indefinite specifications, is still dissatisfied with what he receives.

The consumer, after spending much time and money testing coals in order to find the best fuel value, based on results and costs in his plant, still feels the need of definite standards by which both inherent and preparation values can be readily specified and measured, in order to be sure of getting the results shown by his tests.

In the lumber industry, where the many kinds, sizes, grades and uses would seem to offer insurmountable difficulties, standards have been adopted which work to the satisfaction and benefit of all concerned.

Unquestionably, there are many complications and difficulties to be solved before such standards can be made fully effective in the bituminous coal industry, but if a start is made, there is no doubt that standards can be worked out which will gradually come into general use; and which will be of great benefit to the individual producer and consumer, and to the industry as a whole.

DISCUSSION

(*John J. Lincoln presiding*)

MEMBER.—Why have not the New River and Winding Gulf fields made the same progress along this line as the Pocahontas field?

P. C. THOMAS, Pittsburgh, Pa.—The New River field led in the adoption of modern screening plants, but the rapid increase in the sales of smokeless prepared sizes has encouraged the erection of numbers of excellent preparation plants in both fields. I would say, for the amount of money invested and the size of the plants, the Pocahontas field probably leads at this time. Mechanical preparation, such as dry cleaning and washing, is more prevalent in the Pocahontas field, probably because of the bone that occurs in the No. 3 Pocahontas seam. The Sewell seam generally is clean, yet there have been a number of mechanical cleaning plants installed in that field during the past two or three years. There has also been considerable activity in modernizing tipples in the Winding Gulf field, using both wet and dry methods.

C. SCHOLZ, Charleston, W. Va.—I came to West Virginia in 1920 to develop two coal mines, one of which is Glen Rogers operating in the Beckley seam. After careful consideration I concluded to build a three-track tipple with shaker screens, for two reasons: (1) to be able to hand-pick the coarser sizes if the coal had to be reassembled should it be necessary to sell it as mine run and (2) to get the advantage of higher prices for prepared sizes. My principal complained very bitterly, believing that all smokeless coal should be sold as mine run, and quoted E. E. White as agreeing with him. Mr. White had just completed a new tipple at Glen White, W. Va., which could handle mine run only. Now after 10 years the Glen White tipple has been rebuilt and equipped with a very complete screening outfit. Glen Rogers had to add additional units of screens and now makes five different sizes.

R. F. ROTH, Fairmont, W. Va.—Mr. White began screening the Beckley coal at Stotesbury in 1913. He began operating in 1910, and began to market screened coal as soon as the mine was fully developed. At Glen White it was different. Mr. White himself made the decision not to install screens at that plant on account of the peculiar structure of the Beckley seam. The full seam was worked for many years at Glen White and extended over a large area. The upper bench was hard and the lower bench was very friable, so that the percentage of slack was extremely high, to say nothing of a marked difference in the analysis of the separate benches. Mr. White also considered it good business to dispose of the Glen White coal as a run-of-mine product as long as he could get a good price and make money without going to the expense of installing screens and then penalizing himself under the conditions as they existed at that time. This may explain the apparent 10 years' lag.

E. O'TOOLE, Gary, W. Va.—Mr. Guy did not give the date when the cleaning of coal by water started, neither did he state when the cleaning of coal by air began. The air process is new—about 10 years old.

There are several dry-cleaning plants on Winding Gulf. When necessity arises there will be cleaning plants in the tipple, and when there is no necessity it would be a waste of money.

The day is coming soon when it will be necessary to ship the best coal possible. And when we speak of clean coal it will mean coal from which no more dirt can be taken.

The consuming public has been taught that washed coal is clean, which is generally true, but no attention is given to the fact that in washing water is added to the coal, which increases the weight and freight. The freight on coal is the big factor—not the cost of the coal at the mine—and for that reason a number of operators who are now cleaning coal by water are considering cleaning it by the dry process.

Bituminous coals in their natural state contain water in various amounts, and in various conditions. There is the surface moisture—moisture from the atmosphere and from the seam—on the surface and in the pores of each particle of coal exposed, and there is water of crystallization—hydrogen and oxygen that is crystallized in the coal in a hygroscopic form, variously called inherent moisture, water of crystallization, hygroscopic moisture and water of hydration. It is water or moisture, no matter by what term designated.

Very little, if any, water is encountered in mining horizontal coal seams, except where the surface is broken, but where the workings are to the dip of the seam, water accumulates. Most coal mine workings are to the rise, and are dry; so dry that in some states the mining departments insist upon water being sprinkled on the cutter chain of the mining machines as they undercut the coal, and around the loading machines as they load the coal, and around the places where explosives are used. In some instances they insist upon sprays being set along the haulage roads so that the dust on top of the loaded mine cars will be well watered. Where coal is dumped from the mine cars into skips and elevators or on to belts in the mine, they insist upon water being sprayed upon the coal as dumped, as coal dust in suspension and mixed with a proper proportion of air is explosive. All such watering adds probably 4 to 5 per cent of surface moisture to the coal coming from the mine.

When this wet coal is passed through dry cleaning separators for the purpose of removing the separable dirt, a portion of this moisture as well as the separable dirt is removed, leaving the coal in its naturally dry state.

The removal of 2 per cent moisture from coal when it is being cleaned is equivalent to reducing the freight rate on the usable coal at least 3 per cent, as it will require 1 per cent of the coal shipped to vaporize the 2 per cent moisture it contains. The removal of 4 per cent separable ash or dirt from the coal is equivalent to a saving of approximately 6 per cent in freight on the useful coal, as the separable ash or inert matter which the coal contains when consumed will carry off and render useless the heat produced by one-half its weight in coal.

The high freight rates increased the importance of shipping clean, dry coal. Expensive coal gives oil and electricity an additional advantage, and the day is here when it is impossible to market at a profit anything but coal that is both clean and dry.

The cleaning of coal by the dry-cleaning method reduces the moisture and the ash by removing the water and separable foreign matter. The removal of the foreign matter reduces the sulfur and phosphorus, as the separable dirt carries a large portion of these elements. To remove the separable ash and moisture in coal increases the percentage of the volatile hydrocarbons, the fixed carbon and the fusing temperature of the remaining unremovable ash.

Many coal producers have failed to keep up with the parade of progress. Some are still shipping their coal run-of-mine and when they do size it sometimes they do not clean it, and are still advocating that even clean coal should contain a certain amount of permissible high-ash coal (bone) and a certain percentage of slate, etc.

The high cost of freight to the coal consumer has caused him to devote his energies to the more scientific use of coal to keep his fuel cost within reason. The coal producer will be forced to meet this condition by shipping only coal that is properly sized to the service for which it is intended and absolutely free from all slate, water, etc.

G. R. **Delamater**, Cleveland, Ohio.—At the bottom of page 57, Mr. Guy says: "It is striking that the same mines that installed mechanical cleaning equipment from 20 to 28 years ago produce a greater ratio of the tonnage from the field in 1930 than in 1910." This should be food for thought to all interested in this subject.

Another most enlightening statement is made on page 58, indicating that this group of mines produced 14.4 per cent of the Pocahontas field production in 1920 when prices were high and when there was a great demand for coal, and 17.9 per cent in 1921 when there existed a severe business depression.

I am much interested in the author's remarks regarding the need of standardization. A common complaint of the coal operator against mechanical cleaning of coal is that no increase in price can be obtained to compensate such cost of cleaning. In September, 1930, David R. Mitchell said, regarding the coal cleaning situation in the State of Illinois:[1] "The washing of coal has almost entirely disappeared from this state. This is a rather startling fact when one considers that in the year 1908 Illinois outranked all other states in quantity of coal washed and a few years later, in 1912, 85 washing plants were reported."

Most coal consumers seem interested only in purchasing coal at a minimum price, quality being of secondary importance so long as the general character of the coal suits their particular requirements. Many of them realize to some extent that it is more profitable to use coal of a constant ash content, but they do not seem to realize the cost to the operator to provide such coal, and the operator has done little to convince the consumer that such an attitude is in fact a boomerang.

Frequently fear is expressed that gas and oil jeopardize the coal industry. The gas industry is strongly bonded together by the American Gas Assn., which is directed and controlled by the gas-producing companies, not by any organization whose interest is general, as is that of engineering societies. The entire purpose of the American Gas Assn. seems to be to promote the interests of the gas producers. The work of their committees surpasses that of any other organization of this general character, and their reports cover every conceivable thing that could possibly affect the gas industry, either beneficially or detrimentally, and that means among other things that they know all about the coal industry and its possible effect on their own future. They support and maintain an elaborate and complete laboratory in Cleveland and a branch in Los Angeles, where thousands of dollars are spent in research that will protect and better the future of the gas industry and in the development of equipment for the same purpose.

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[1] *Trans.* A.I.M.E. Coal Div., (1931) **94**, 301.

The Association has adopted an insignia now universally accepted as a stamp insuring both safety and efficiency of any equipment upon which it appears. A manufacturer of gas-consuming equipment has difficulty selling his product unless that insignia appears prominently thereon, and the use of this insignia is only possible by submitting such equipment to the A. G. A. laboratories for exhaustive inspection and test, where it must meet all of the carefully established standards of perfection, the object of which is to assure the gas-producing companies that the reputation of gas will not be injured through inefficient or improper use.

The entire set-up of the American Gas Assn. is such that the gas-producing companies, the gas-consuming equipment manufacturers and the gas consumers are brought into closer relationship, and this effects greater cooperation between them.

While there are both local and national associations of coal operators, there are none patterned after the American Gas Assn. They do not provide financial support for a well equipped and well organized research laboratory operated under their own supervision and control.

For many years there has been a steady increase in the number of by-product coke ovens in this country. Much of the surplus gas from these plants is consumed for domestic purposes. The coal used by these plants is probably consumed more efficiently than coal used for other purposes.

While gas as a competitor of coal, and increased efficiency in the use of coal, may have something to do with the present unsatisfactory conditions in the coal industry, they are natural developments and it is a loss of time and energy to combat them. Natural gas requires no consumption of coal in its production, yet at periods of heavy demand water gas and coke-oven gas frequently must make up the deficiencies of natural gas.

Many, if not all, states fix by law the minimum candlepower and B.t.u. value of gas sold for public use. The consumer of coal is not so protected as regards the sale of coal of inferior grade. While such laws pertaining to the sale of coal may at first thought seem dangerous to the coal industry, there is considerable possibility that they might be drawn so as to be extremely beneficial. Both the producer and the retailer are seriously affected by the sale of inferior grades of coal, particularly when sold by irresponsible companies or individuals with no overhead or permanent equipment. Properly drawn state laws might eliminate both and automatically close down permanently the poorly equipped mines with inferior coal which flood the market and undermine the legitimate price.

Perhaps this is off the subject of standardization, but with a national coal producers association patterned after the American Gas Assn. is it not possible that much could be done that would standardize not alone the preparation, but also the production and the consumption of coal, and have much to do with the provision of state laws that would be a great benefit to the entire coal industry?

Mr. Guy cites the need of "cooperation of producers and consumers in establishing standards for the preparation and marketing of coals." One difficulty is that many of the consumers of coal are individuals who cannot be brought into cooperative organizations. They know nothing of production or preparation. They understand only price. They are protected by law against short weights only. Laws protecting them against inferior quality would be their indirect cooperation in establishing standards of preparation and marketing of coals. Such laws would of necessity provide different standards for different uses, even to the extent of prohibiting the improper use of coal. For example, a well prepared but high-volatile coal is of great value for certain purposes, but it is almost criminal to sell it to the unsuspecting household consumer and is also an almost criminal wastage of one of the coal industry's most valuable products.

Mr. Eavenson, in speaking of the use of high-volatile coals for domestic purposes,[2] reminded me of an occurrence in Cleveland about two years ago, which I believe hurt the coal industry a great deal. An advertising campaign was put on to advertise a clean washed coal. The company that advertised had a good reputation, so its statements were accepted by the public. The coal was washed, but was an exceedingly high-volatile coal. Many people filled their cellars with it, and I knew some who bought this coal who had to take it out of their cellars because they could not keep their chimneys open. The fact of the matter was that this was a splendid coal for certain purposes, but the sale of that kind of fuel to some consumers, household users, for instance, hurts the entire coal industry.

L. N. Thomas, Carbon, W. Va.—This discussion has been limited entirely to the practices and experiences of the operators of the Kanawha field, mining largely in the Dorothy and Coalburg seams. Two paragraphs in Mr. Guy's paper are particularly significant; the one on page 60 beginning "The close coordination of preparation and marketing plans," and that on page 61 beginning, "In the absence of standards." While the problem of establishing standards in the preparation of coal will eventually become a complicated problem when carried to points of great refinement, there is a great deal of useful work that can be done without presenting at the start any serious difficulties. This may be grouped roughly under two headings: (1) classification as to size; (2) classification as to foreign matter or impurities.

Under the first head, since shaker screen equipment has so long been in general use, it may be a surprise to many that so much misinformation and lack of understanding can exist as does exist between the trade and those in charge of the preparation at the mines. Domestic coal is sold as block, egg, nut, etc., or as 4-in. lump, 2 by 4-in. egg, and 1 by 2-in. nut. Several companies may be advertising under these same terms, yet the 4-in. lump of one company may be the product which passes over a 4-in. round opening; that of another company may pass over a 2¾ by 11-in. lip screen. The egg of one company may pass over a long-slot lip screen and through a round opening, while the egg of another company may pass over a round opening and through a short-slot lip. These screens may supposedly be theoretically equivalent, yet, owing to this difference and the methods of shooting and handling, and the cleavages of the coal, the 2 by 4-in. egg coal from one company will appear quite different from that of another operating in the same seam.

The matter of tolerance on "oversize" and "undersize" might also come under this heading. Under some methods of handling, any coal encountered in the seam, which is of a softer structure than the general run, is broken up before reaching the picking tables, and, therefore, is screened into the smaller sizes where breakage in transit after loading in the railroad cars is not an important consideration. In other systems the coal is handled to avoid breakage and the hand pickers are relied upon for the removal of the coal of softer structure which has passed over the screens into the larger sizes. The more modern outside plants appear to be designed along the latter lines.

Sometimes two cars of coal loaded from the same mine will contain quite different quantities of the fines resulting from breaking in transit, although in both cases the amount may be negligible. Concessions to the trade result—the amount depending upon the standing of the complaining party and the severity of the complaint.

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[2] See page 96.

There appears no good reason why specifications could not be adopted designating the size and shape of screen openings for each size or grade of coal, and for the allowable quantity of undersize.

Under the second heading, standards of impurities, even more can be done to eliminate the never-ending controversy between trade, sales and operating department. When competition is keen it seems to be the ever-increasing policy of salesmen from the various companies to console a dealer, whose grievance is a small pile of 50 lb. of slate in a 50-ton car of coal, by making him a present of 25¢ a ton on the contents of the entire car. At the same time, the operating department, after weeks of careful study and planning under the stress of the times, is congratulating itself upon arriving at some plan which lowers its cost 3¢ per ton. The salesman justifies his action by the firm belief that this quantity of impurities is excessive and that if some such action is not taken the privilege of consoling the dealer will go to his competitor who will guarantee a car of coal containing no foreign particles. The operating department is severely roasted and requested to investigate and report the causes, which of course cannot be done. If the competitor gets the business, the chances are this experience will be the same. If a representative of the operating department tries to explain that 50 lb. of slate is not excessive, his chances of success depend upon the dealer's fairness and state of mind at the time of his arrival upon the scene and the dealer's state of mind may be influenced largely by the fancy promises made by another salesman. The salesman does not know how much slate should be allowable without complaint and neither does the dealer.

Why could not some standard quantity of impurities be set up in the prepared sizes of coal and a definite ash content on the fines, which would govern both operators and dealers. The specifications for ash could probably be made uniform for one seam wherever it occurs. Standards for screens could either follow this or be made to cover certain fields. Standard methods of sampling fines should be adopted. These details could be worked out by a committee of operators familiar with the various seams of coal and the dealers familiar with their uses. They may not at first be exactly clear but it seems to us that the benefit to be derived is quite plain.

H. N. EAVENSON, Pittsburgh, Pa.—The movement for standardization of clean coal was started by some operators who were cleaning their coal, in response to demands for a definition of clean coal, and after a discussion we took the matter up with the Institute and assumed the responsibility of obtaining approval of an American Standard by the American Standards Association.

In the first place that body had to decide whether it wanted to take it up or not, and asked us to prepare a statement of what we wanted to do. We did this and it was referred to the Association. I sent a copy of Mr. Guy's paper to each coal company in West Virginia and Pennsylvania, and I got the impression from them that they thought that we were trying to standardize the size, and the National Coal Association thought the same thing and was opposed to it on that account.

I doubt whether standardization of size can be done for soft coal. It is entirely different from hard coal. It may be put into the cars at the mines in good shape, but when it gets to the consumer it will be smaller.

I discussed the matter further with the American Standards Association, and with some who were interested yesterday, and am going to change the scope of the project to read as follows:

STANDARDS FOR CLEAN COAL.—Specifications outlining allowable limits of inacceptable material in any size of prepared bituminous coal, whether mechanically cleaned or not; the purpose being to define "clean coal" and not to standardize screen sizes.

We are going to send this if approved to the American Standards Association. Our idea was to define clean coal. It might be high ash and clean, and low ash and clean, and without any preparation.

G. S. Rice, Washington, D. C.—One or two queries have occurred to me regarding the question of what is "clean coal," which probably Mr. Eavenson can answer. Let us go away from this Pocahontas field with its high-grade coal and consider the question of clean coal in other more difficult districts, say in the State of Washington, where the different layers of bituminous coal in the same bed may vary from low specific gravity up to coal that may have a specific gravity of 1.5 or more. Suppose the washing is done as completely as is practical. What is to constitute "clean coal"? If it is to be "pure coal" it might result in the rejection of 50 per cent of the washery product that could be and is used as fuel.

And then comes the question of conservation, where the specification of clean coal is made severe. When the Bureau of Mines, at the request of the navy, 20 years ago, made a thorough fuel survey, with sampling, of all the mines in the Pocahontas and New River fields it was found that to meet the navy specifications the operators had to reject thousands of tons of coal carrying higher ash, yet which would have been considered good fuel coal in other parts of the country.

To meet satisfactorily such difficult questions, it seems necessary that specifications applied nationally for "clean coal" should have flexibility to fit different coals for different purposes. I should also like to hear from Mr. Eavenson as to how it is proposed to define extraneous impurity of coal in distinction to inherent ash and moisture.

H. N. Eavenson.—A committee would have to work that out. It could take the coals east of the Mississippi. I do not see why national specifications could not be made, because the term "clean coal" has no reference to inherent ash. The committee would have to say whether it considered inacceptable material as bone or slate or rock. It would be easily possible to make a specification for this portion of the country and a separate one for Washington. Most of the coal is produced east of the Mississippi River, and a specification for that would cover a large percentage of the whole territory.

A great deal of the coal taken out of this field and thrown away is better than 95 per cent of the coal in the State of Illinois. Here we cannot sell it or give it away. We cannot use it now, and under present conditions of the coal business I do not see any prospects of using it in the near future.

I do not think anybody knows what "clean coal" is. You may put out the very best you can, and somebody will come back and say he found 40 or 50 lb. of bone or slate in a car, but what is that in a 100,000-lb. car?

I believe that if a standard were prepared and the people educated up to it—say 100 lb. to the average car—we could say that there was no ground for complaint.

The specifications for anthracite mean something, and are lived up to. The same situation could be carried out with bituminous fields. Now, if a man says he found a certain amount of slate in a car you are obliged to admit it, and the easy way to get out is to make an allowance.

G. E. Bayles, Nemacolin, Pa. (written discussion).—The aim in preparing coal is to attain uniformity in all of the characteristics of the resultant product, physically and chemically, as well as the elimination of a maximum of removable foreign material. Under any except abnormally good market conditions, coal that is not uniform in its quality is losing out, and is being replaced by a properly prepared product. The inherent characteristics of bituminous coal, in general, vary to such an extent that only general standards of quality can be applied to the production from any seam, district or field. Standards of grade, size, percentage of bone and slate, and of fuel value, should be set up as a base line, and a definite scale of premiums and penalties established for super or substandard coals.

The problem then facing the producer, in the preparation of his coal, is the economic problem of balancing the degree of elimination of foreign material, with the premium that such elimination will gain. If, knowing the standard, its premiums and penalties, a producer overreaches himself or fails to attain the economic limit in the preparation of his coal, he has no one to blame but himself for the resultant heavy loss. There should be complete cooperation between consumer and producer in establishing the standards for preparation of coal, so that they may be effective, and that the benefits may be mutual.

J. Daniels, Seattle, Wash. (written discussion).—While I believe in the general principle that uniformity in quality or grade of coal shipped is extremely desirable, nay even necessary, as one of the steps to stabilize the coal industry, I am not convinced that any one set of standards can be drawn up to cover all the coals marketed in the United States. This is particularly true of our western coals, and especially so in the case of coals mined and prepared in Washington. It might be possible to set up regional or district standards, however, that would function successfully.

Uniformity in sizes and designation of market sizes is possible, and it might be found feasible to standardize these as a beginning.

H. F. Yancey, Seattle, Wash. (written discussion*).—Mr. Guy has presented a timely discussion which shows that definite advantages accrue to the producer of mechanically cleaned coal, particularly in times of a buyer's market. Uniformity of product with respect to both size and quality is the most desirable and important attribute of a well prepared coal, and such uniformity is characteristic of the product resulting from mechanical cleaning plants operated under technical control.

Although the adoption of equitable standards governing the quality of prepared coal may appear uninviting and the standards themselves generally difficult to formulate so as to conform to the views of all the operators in a given field, it must be apparent to the operators that consumers have been proposing their own standards for a long time. The purchase of coal according to a specification covering ash content, heating value, etc., may be regarded as a process of standardization by the consumer, and the requirements specified in such contracts are often those maintained by some one of the more carefully operated cleaning plants in the field. If the operator and consumer do not cooperate in formulating standards, the consumer will formulate them alone. Obviously, such a solution cannot be as uniform in basis or as satisfactory to all concerned as one resulting from the joint action of the producer and the consumer.

The adoption of standards in a given field should be based upon preliminary work done by the operators themselves. The first step would involve the determination of the characteristics of each of the sizes marketed at all the mines in the field. This work should provide a basis from which tentative standards could be arrived at. Such preliminary work should be done according to a well thought out and considered plan and should not be undertaken until such a plan is developed and agreed upon. Some of the questions to be decided in formulating such a plan are mentioned in the following paragraphs.

The weight of the gross sample to be collected for each size of prepared coal must be decided, and likewise the manner of collecting the gross sample, with particular reference to the weight and number of portions included and the period over which the gross sample is collected. If the percentage of high-gravity material contained in the market sizes is accepted as one of the characteristics to be determined, as has been suggested in previous discussions before the Institute, the use of the precise float-and-sink method of testing is advocated instead of the arbitrary visual-inspection method.

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It is obvious that the determination of the percentage of high-gravity material contained in a given product does not, by itself, determine the quality of the product. One must also know the percentage of low-gravity material in order to know the quality. It is conceivable that a product containing no high-gravity material may be of very inferior quality; for example, it could consist almost wholly of bony material of intermediate density. Perhaps the better way of insuring the quality would be to determine the ash content of the sample and the percentage of high-gravity material present. This would eliminate the determination of the percentage of low-gravity material.

It seems evident that different standards would have to be set up for different fields or different mining districts. In order that these standards should prove equitable for all the operators in the field, and therefore beneficial to the producers and consumers alike, they should be formulated as the result of experimental work done by or under the direct control of the operators' association in a particular field or district.

R. E. **Rightmire**, Fairmont, W. Va. (written discussion).—In his remarks on standardization the author covers broadly the urge for and advantages of such an arrangement for bituminous coal. Practical uniformity in the marketed product is generally recognized as a highly desirable and valuable feature both to the producer and consumer.

The physical and chemical character of bituminous coals, both high-volatile and low-volatile, vary so widely that a set standard for them, in general, or even for a particular locality, must necessarily be so broad that the marketed product, in most instances, would not be defined closely enough by the standard to satisfy either the producer or consumer. Also, the consumers' plant-operating conditions vary so widely that discrimination within the standard must necessarily be practiced to satisfy the different operating requirements and in that event it still remains for the selling agent to define his product to the customer with the necessary detailed information, and the producer's responsibility in making certain that shipments meet satisfactorily the customer's particular requirements would remain unchanged.

Standardization with respect to preparation would cover the extent of removal of free impurities and also size and screen sizing, but for the last feature it probably would be necessary to provide for making a multiplicity of screen sizes, because of established market demand in different localities and the natural resistance to change of long-established practice. This standardization should be fairly readily accomplished, dependent upon the extent of cooperation of producer and consumer.

Standardization with respect to analytical quality, character of ash and fusion temperature offers little or no encouragement, because of inherent characteristics of the raw products which are not amenable to control. There is the further difficulty of harmonizing these features in a standard to represent both hand and mechanical preparation.

Standardization that would represent fairly closely the product as shipped, and not tabulated information as to the average of wide range in character and quality of the coals, appears highly desirable, and regardless of the many difficulties to be solved in accomplishing this further study and investigation of the problem appears to be well warranted.

H. F. **Warden**, McComas, W. Va. (written discussion).—The coal industry is greatly indebted to Mr. Guy for his able presentation of the subject of his paper.

It is noted that the first washer in the smokeless fields was installed at the Crane Creek mine in 1903 and the first commercial installation of pneumatic cleaning equipment east of the Rocky Mountains was made at this mine in 1923. A new washing and dry-cleaning plant was completed the early part of this year. Exceptionally fine results have been obtained in preparing and cleaning the small sizes for a market which requires and demands uniformity of size and quality.

The coal produced is from the No. 3 Pocahontas seam, averaging about 4½ ft. in thickness. Impurities consist of several inches of rotten slate or "muck" and one inch of 35 per cent ash cube coal at the top of the seam and the characteristic bone and "sulfur band" partings occurring in this seam. The coal is machine cut and hand loaded, every effort being made to clean the coal as much as possible at the working face. The preparation plant operates 14 to 16 hr. and prepares 4000 tons daily. Sizes are lump, egg, stove, chestnut and slack.

Lump is screened out on a marcus screen, hand-picked, rescreened and loaded over shaker loading boom into car. Lumps removed from picking table containing bone are crushed through single roll crushers, crushed material being mixed in with the crude coal coming from the mines. Egg is screened out on marcus, hand-picked, rescreened and boom-loaded into car. Refuse elimination from lump and egg grades amounts to 6 per cent and consists of pure bone and slate. Stove is screened out on marcus, lip-screened and washed in Menzies hydroseparator. Clean coal is dewatered over wire screen, carried by belt to loading point where rescreened and loaded direct to car. Refuse is carried by scraper line to refuse bin. There is 9 per cent refuse rejection from this grade, consisting of bone and slate and 55 per cent of ash in refuse.

The lump, egg and stove sizes are used altogether for domestic purposes and after the treatment outlined above are consistent in preparation and good quality. Nut and slack, which is 56 per cent of total output, is screened through marcus screen into bin, from which it is elevated by rubber belt to a Parrish screen, the slack screens through, the chestnut going direct into Menzies hydroseparator. The clean chestnut is dewatered over wire screen, delivered to belt and transported direct to car. Refuse goes by belt to refuse bin. The crude chestnut runs 10 per cent ash; the cleaned, 6 per cent and refuse 51 per cent ash. There is 9 per cent refuse rejection, consisting of bone, slate and cube coal.

Slack coal at this plant is difficult to clean because more than 50 per cent is below ⅛-in. ton cap and the impurities are very friable, being found in about the same proportion in all intermediate sizes between ⅝ and 0 in. It becomes necessary to clean efficiently down to 20 mesh or below, in order to secure a satisfactory product. To meet the requirements the ⅝ to 0-in. coal is air-cleaned on primary tables and the below ⅛-in. product screened out and retreated on secondary tables.

The crude slack, 150 tons per hour, is distributed to four American pneumatic separators of the more recent type and increased size. Fans pull the fine dust out of the coal as it comes on the separator. The dust is drawn through piping to an American tube filter, which collects it and delivers to screw conveyors, which in turn empty on to a clean coal belt, mixing back with the dry-cleaned slack. As the ⅝ to 0-in. crude passes over the separator all impurities are removed between ⅝ and ⅛ in. and some below ⅛ in. The product is passed over ⅛-in. Hum-mer screens located under each separator. The clean ⅝ by ⅛-in. over product delivers to the clean coal belt and the through ⅛ to 0 in. is elevated by bucket elevator and distributed to three additional American separators. All the remaining impurities between ⅛ in. and 20 mesh and some between 20 mesh and 0 in. are removed by these separators. The cleaned ⅛ to 0-in. product is delivered to the clean coal belt. Refuse from all the separators is delivered to belt and carried to refuse bin.

The crude ⅝ to 0-in. slack goes into the separators at 8 per cent ash and comes out clean at 5.5 per cent ash. The refuse runs 40 per cent ash. There is 7 per cent refuse rejection from this grade.

As a result of the application of the air cleaning process, the Crane Creek plant is able to deliver, especially for metallurgical purposes, a consistent, low-moisture, low-ash slack coal.

To raise the standard of preparation of all coal to the point where the consumer gets in each and every ton a satisfactory product should be the goal of all persons connected with the coal industry.

T. Fraser, Pittsburgh, Pa.—There can be no doubt that something is to be gained by airing this question of standards of fuel quality, whether or not we succeed in setting up workable quantitative standards. The movement to set up standards of coal quality is a manifestation of the trend toward purchase of coal on specification, which is the ultimate objective. The adoption of definite standards and cataloguing of available coals by quality rating would enable the small user as well as the larger user to buy coal on a basis of fuel value.

Closely associated with preparation standards is the subject of guarantees covering the performance of coal-cleaning plants. This question, involving the solution of an equation with several unknown and variable factors, is one that the plant seller would like to have more widely understood.

The quality of a shipped coal product depends very largely upon conditions fixed by nature and often to only a minor degree upon preparation; hence the so-called standardization program must be made to fit the coal rather than depend upon making the coal conform to the standards. The standardization schedule will therefore be a compromise between classification and standardization and may well consist of two steps:

1. Classification of coals according to inherent quality and preparation characteristics.
2. Standards of preparation adapted to each group in the classification.

In this connection I would call attention to the suggested scheme of classification according to washability characteristics presented in U. S. Bureau of Mines *Bull.* 300, in which American bituminous coals are separated into seven broad groups identified by differences in physical form of impurities. Some such classification as this extended to cover such preparation factors as friability and particle shape might form the base for a flexible standard of preparation.

D. R. Mitchell, Urbana, Ill. (written discussion).—Mr. Guy opens up anew the question of benefits to be derived from standardization of bituminous coal. Standardization seems to go in cycles. During periods of poor business considerable interest in it is shown as a means for stabilizing the coal industry, but unfortunately progress gained during such times is frequently lost during boom times, when anything that is black can be sold for coal.

The arguments pro and con are well known. Mr. Guy has enumerated exceedingly well the benefits that would be derived from the adoption of a set of standards for the smokeless fields of West Virginia. The chief arguments against adopting standards are that the great differences in quality, rank and usages of bituminous coal are of such a nature that it would be impossible to adopt a set of standards that would apply to all bituminous coal fields and for all fields of use. I do not believe that such objections are insurmountable, rather that they point the way to which adoption of standards should be attacked; namely:

1. The recognition that problems of bituminous coal preparation standards are regional and not national.

2. The recognition that the coal dealer and coal consumer are as vitally interested as the coal producer.

Regional committees representing the chief producing coal areas could be appointed to make a report on the possibilities in their respective fields, limiting their initial reports as far as possible to the simpler aspects of the problem such as sizing, degradation, limitation of free impurities and bone. Further research work is needed in most fields to adopt any very definite standards based entirely on chemical analysis and methods of use. This is true of most of the bituminous fields. Central Pennsylvania and the Illinois field, for example, probably would require a set of standards for each coal bed worked in each county in the respective fields.

Any movement furthering the standardization of bituminous coal in the various producing fields will have my hearty support.

L. E. Woods, Huntington, W. Va. (written discussion).—In discussing the question of establishing standards for coal preparation in the bituminous field, the subject might well be considered in three distinct phases:

1. Standardization as to quality with particular reference to the allowable limits of extraneous foreign matter in any size of prepared coal.
2. The maximum amount of oversize and undersize allowable in each size.
3. Standardization of sizes.

With reference to standardization of allowable limits of extraneous foreign matter in any size of prepared coal, inasmuch as some seams of bituminous coal are practically free from any extraneous foreign matter, those coals automatically fix the standard, through competition, as to the amount of extraneous matter allowable; and those who are not fortunate in having clean seams are compelled by competition to remove all the foreign matter possible in order to compete. This is particularly true of the sizes used for domestic purposes.

While the anthracite region has set up a standard fixing the amount of extraneous matter allowed in the various sizes, all seams of anthracite contain more or less foreign matter and a standard can be set up; however, the liberality of the standards adopted in years gone by has probably caused the loss of considerable anthracite tonnage to the bituminous industry, and I understand the present tendency is to produce coal as nearly free from extraneous matter as possible.

In the smaller sizes of coal used for steam and metallurgical purposes, where both quality and uniformity of product are advisable, due to the wide variation of inherent noncombustible matter in various bituminous coals, any attempt to stabilize quality in the finer sizes should consist of fixing a differential in value as between the various qualities of coal produced rather than the foreign matter allowable, so as to encourage the purchase of these coals purely on the basis of quality.

I doubt that standardization is feasible in regard to the maximum amount of undersize and oversize allowable in any specific size, because of the wide variation in the structure of bituminous coals. The amount of undersize and oversize in coal prepared in modern tipples today is so small that it would be useless to fix a standard. By the time the coal reaches the customer the amount of oversize and undersize depends almost entirely upon the structure of the coal, breakage in transportation and handling in unloading—all of which are beyond the control of the producer. Here, again, competition fixes the standard.

There is great need of the standardization of sizes in all bituminous coal. Considerable progress has already been made in this respect in the smokeless industry, but so far no attempt has been made toward standardization of sizing in the high-volatile industry, with the result that competition in sizes produced is almost as keen today as is the competition in prices, and each concession in size made by an individual operator is forced upon his neighbors and competitors. I recently attempted to ascertain some of the standard sizes in use in the Logan and Williamson fields of West Virginia, and find that the standard screens used by 51 separate mines in those two fields are as shown in Table 3. These sizes do not take into account the many special sizes which the mines are called upon to produce.

TABLE 3.—*Standard Sizes of Screens Used in Logan and Williamsport Fields, West Virginia*

LUMP SIZES	MINES	EGG SIZES	MINES
Over		Over	
6-in. round hole	2	3-in. through 6-in. round	1
6-in. slot hole	1	2½-in. round through 5 by 8 in.	1
4-in. by 10-ft. bar screen	1	2-in. by 10-ft. bar screen	1
5-in. slot	1	2½ through 4 by 6 in.	2
5-in. round	2	4-in. round through 6-in. slot	3
5 by 8-in. lip	1	2½-in. round through 5-in. slot	1
4 by 6-in. oval	2	2-in. round through 5-in. round	2
4½-in. round	1	2½-in. through 4½-in. round	1
4-in. round	23	2½-in. through 4-in. round	3
4-in. lip	15	2-in. through 4-in. round	24
2¼ by 3-in. oval	1	2-in. through 4-in. slot	5
2-in. round	1	1½-in. round through 3 by 2½-in.	1
		1¾-in. through 4-in.	6
Total	51		51

During the past few years the domestic trade using smokeless coal has been more favorable to the use of smaller sizes until the egg size has become more valuable than the lump. This has encouraged the making of a larger size of egg, thus increasing the percentage of the higher priced coal and making a corresponding decrease in the percentage of lump. So far, the tendency toward the use of the smaller sizes has not been noticeable in high-volatile coals; however, on account of competition, the size of the intermediate grades of egg and stove have been gradually increased without any corresponding increase in price. In fact, this increase in the sizes has increased the volume of the intermediate grades to such extent that it has demoralized the market and these grades are selling considerably below the cost of production. If the American Standards Assn., or the operators themselves, could arrive at some standard in the sizing of high-volatile coals it would be very beneficial to the industry.

The Low-volatile Coal Field of Southern West Virginia

By Howard N. Eavenson,* Pittsburgh, Pa.

(Bluefield Meeting, October, 1931)

The low-volatile, or smokeless, coal field of southern West Virginia is in Fayette, Raleigh, Wyoming, Mercer, Summers and McDowell counties, in the extreme southern portion of the state, and extends into the adjacent counties of Tazewell and Buchanan in Virginia. McDowell County is the largest producer, with Fayette and Raleigh counties next. The total area of the field (Fig. 1) is approximately 1600 square miles, or 1,024,000 acres.

The percentage of volatile matter in the various seams ranges from 15 to 26 per cent. The upper limit of true low-volatile coal is usually considered to be 23 per cent volatile matter; above this point, the coal usually is classed as medium volatile. This limit will be used in this paper.

Geological Position of Seams

The smokeless seams are among the oldest coal seams in the country, as the Pocahontas group of rocks is the lowest member of the coal-bearing series and rests on top of the Mauch Chunk red shale, in or below which no coal seams of any importance occur.

The Pocahontas group, or Lower Pottsville measures, has a thickness of 750 ft. in southern McDowell County, which decreases to about 350 ft. at Prince, on the New River, in Fayette County, 50 miles to the northeast. In this group, at various points, 12 coal beds have been found, of which 4 have been found in workable thickness and are now being mined.

The New River group, or Middle Pottsville measures, is just above the Pocahontas group and varies in thickness from 1300 ft. in southern McDowell County to about 700 ft. along New River. At various places, 16 beds of smokeless coal have been identified in this group, of which 5 have been found in workable thickness and are now being mined. A general columnar section of these strata is shown on Fig. 2. The workable seams in these strata, arranged in geological sequence, are as shown on page 77.

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* Consulting Engineer.

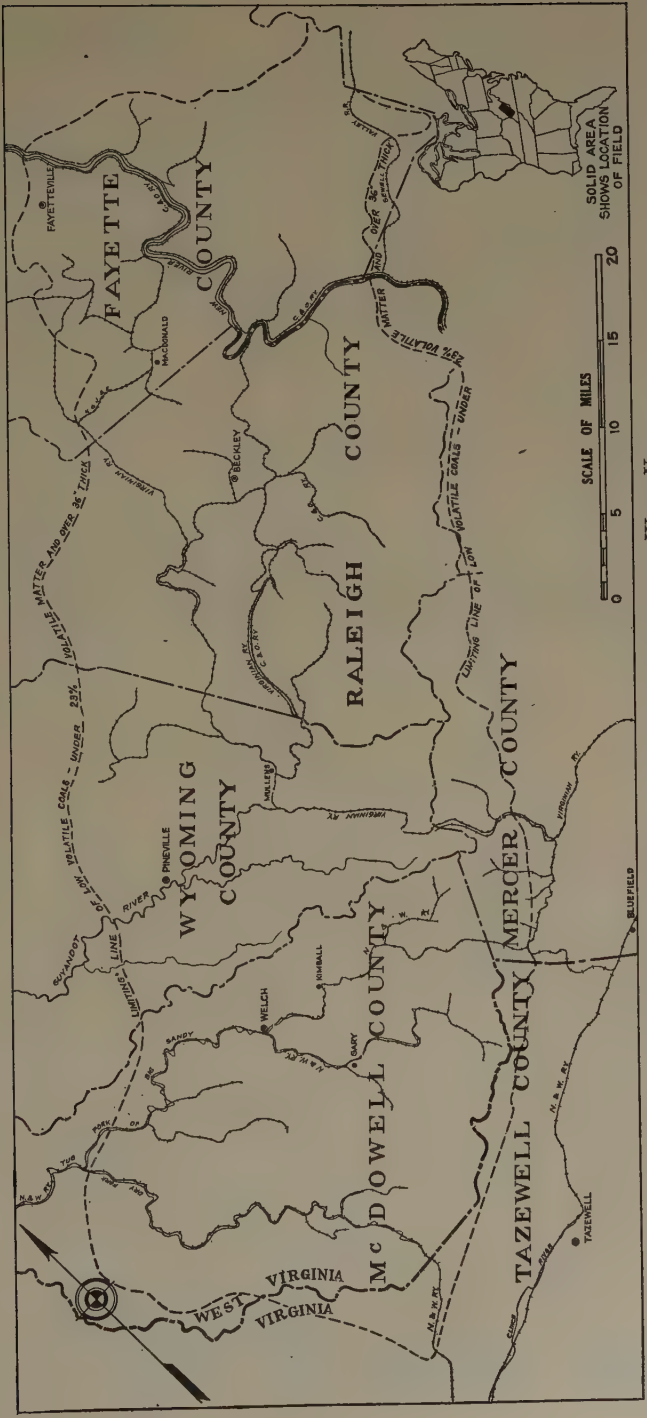

FIG. 1.—SMOKELESS COAL FIELDS OF SOUTHERN WEST VIRGINIA.

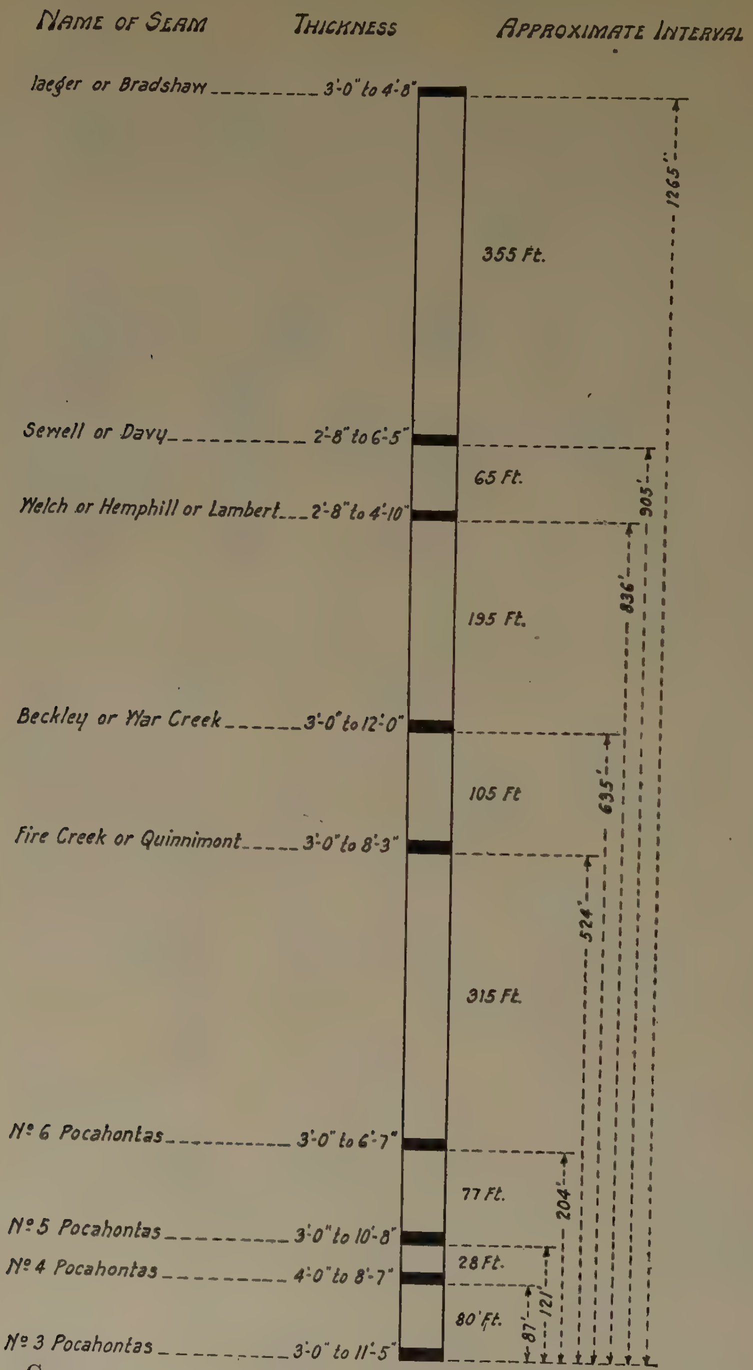
Name of Seam
Thickness
Approximate Interval
Iaeger or Bradshaw 3'-0" to 4'-8"
355 Ft.
1265'
Sewell or Davy 2'-8" to 6'-5"
65 Ft.
905'
Welch or Hemphill or Lambert 2'-8" to 4'-10"
195 Ft.
836'
Beckley or War Creek 3'-0" to 12'-0"
105 Ft
635'
Fire Creek or Quinnimont 3'-0" to 8'-3"
524'
315 Ft.
No. 6 Pocahontas 3'-0" to 6'-7"
77 Ft.
204'
No. 5 Pocahontas 3'-0" to 10'-8"
28 Ft.
No. 4 Pocahontas 4'-0" to 8'-7"
121'
80 Ft.
87'
No. 3 Pocahontas 3'-0" to 11'-5"

Fig. 2.—General columnar section of smokeless coal fields of West Virginia.

Name of Seam	Thickness	Approximate Interval, Ft
Iaeger or Bradshaw	3 ft. to 4 ft. 8 in.	
		355
Sewell or Davy	2 ft. 8 in. to 6 ft. 5 in.	
		65
Welch or Hemphill or Lambert	2 ft. 8 in. to 4 ft. 10 in.	
		194
Beckley or War Creek	3 ft. to 12 ft. 0 in.	
		106
Fire Creek or Quinnimont	3 ft. to 8 ft. 3 in.	
		315
No. 6 Pocahontas	3 ft. to 6 ft. 7 in.	
		77
No. 5 Pocahontas	3 ft. to 10 ft. 8 in.	
		28
No. 4 Pocahontas	4 ft. to 8 ft. 7 in.	
		80
No. 3 Pocahontas	3 ft. to 11 ft. 5 in.	

The seams shown above are all that have been found in workable condition now—by this meaning a thickness of 3 ft. or more of good coal—but it is probable that as the thicker seams are worked out, closer prospecting will reveal places where there will be considerable areas of the thinner seams that can then be worked profitably. At present they are not considered as having any value.

Description of Seams

A brief description of each of the seams follows, beginning at the bottom.

The No. 3 Pocahontas Seam is the one that has given the Pocahontas field its reputation. It was first developed at Pocahontas, Va., where the seam shows its greatest purity and thickness, first shipments being made in 1883. It has its greatest development in Tazewell, McDowell and Mercer counties, occurs in minable thickness in Summers County and has been found in drilling in Wyoming, Fayette and Raleigh counties, and it covers a larger area than any other seam in either series. Fig. 3 shows the area in which it is 3 ft. thick and over. In the vicinity of Pocahontas-Jenkinjones, a considerable area is 10 ft. thick, and in local swamps 16 ft. of coal has been found. The seam dips toward the northwest, and in every direction from Pocahontas it decreases in thickness and the volatile matter decreases from 22 per cent to about 16 per cent, the latter near Welch; the ash increases from about 3.5 to 5.5 per cent and the character of the binder, which always exists near the middle of the seam, changes from a fairly high-ash bone coal at Pocahontas to a carbonaceous sandstone—60 per cent ash—in the vicinity of Welch. The roof of the seam, where it is thickest, is usually a fireclay "clod" on top of the coal,

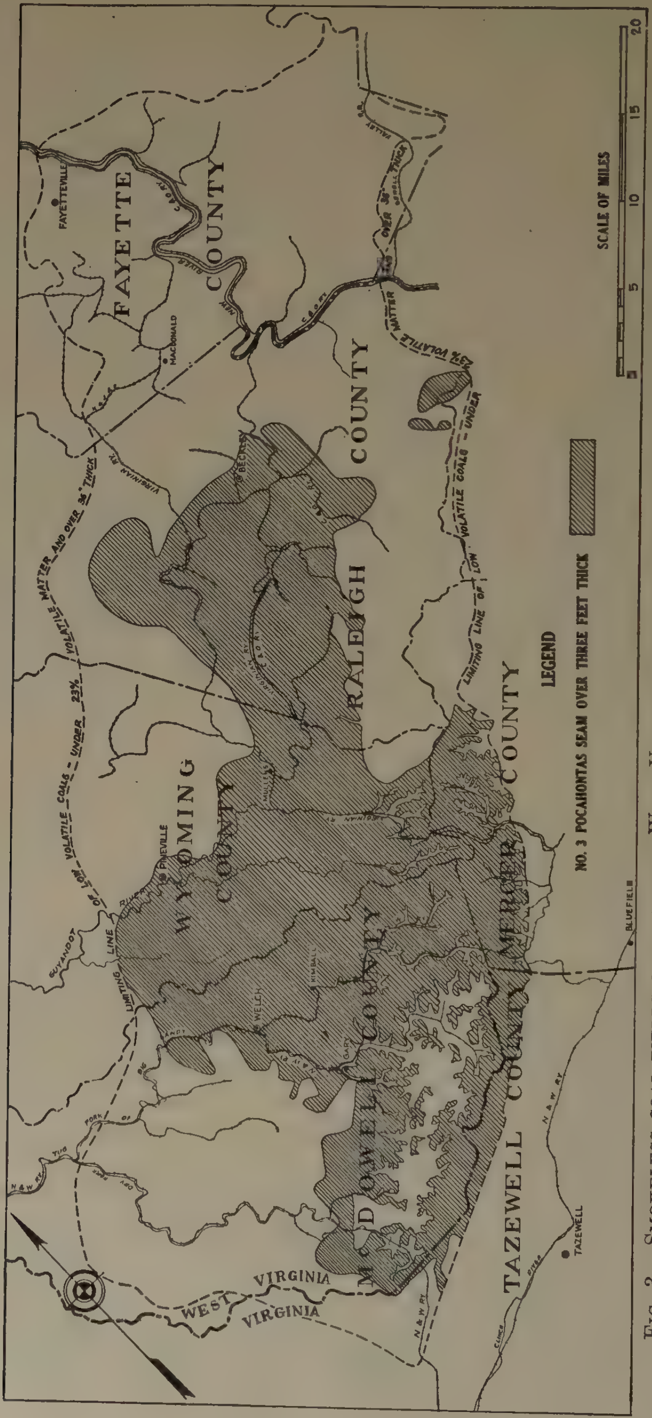

FIG. 3.—SMOKELESS COAL FIELDS OF SOUTHERN WEST VIRGINIA, SHOWING MINABLE AREA IN NO. 3 POCAHONTAS SEAM.

6 to 24 in. thick, which comes down on exposure, but where the coal thins, some parts of the field have a hard slate top which stands well, and in Wyoming and Raleigh counties the sandstone is frequently on top of the coal. In some areas, a small layer of clay, or fine checker slate, occurs, which breaks up into fine pieces and increases the ash in the slack so that it requires washing to find a ready market at all times. In many places a parting locally called the "sulfur band" shows irregularly near the top of the seam. It is rarely over 1½ in. thick and in many places is not found at all. The middle binder is from 1 to 2½ in. thick with the coal "frozen" to it, so that it comes out in pieces about 2 in. thick and can be readily picked out.

The Boissevain fault in Tazewell County marks the southeastern limit of the seam. There are some fairly heavy grades, at one point 15 per cent, in both directions, toward the Dry Fork anticline, beyond which the dip is fairly regular, from 1 per cent to 5 per cent to the northwest. The seam is regular and only one or two very small local "wants" have ever been found.

The condition of the seam where it is over 3 ft. thick, in the various counties, as shown by the County Reports issued by the West Virginia Geological Survey, is shown in Table 1.

TABLE 1.—*No.* 3 *Pocahontas Seam*

County	Number Of Measurements	THICKNESS - INCHES								
		CLEAN COAL			IMPURITIES			TOTAL SEAM		
		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
Tazewell	65	137	36.5	98.3	14	-	3.2	137	36.5	101.5
McDowell	129	113	32.5	65.4	15	-	2.6	128	36	67.9
Mercer	73	127	36	59.3	11	-	3.5	129	36	62.8
Wyoming	40	67	35.5	50.5	13	-	3.8	72	37	54.3
Raleigh	32	92	33	43.4	11.5	-	3.9	93.5	36	47.3
Summers	4	46	36	41.0	4	-	2.3	49	36	43.3
Total Average	343	137	32.5	66.2	15	-	3.1	137	36	69.3

The No. 4 *Pocahontas Seam* has been found in workable thickness only in McDowell, Wyoming and Tazewell counties, as shown on Fig. 4. Its best development is in the area west of Sand Lick Creek of Tug River, where over a large area it is from 6 ft. to 8 ft. 6 in. thick. The seam is chemically about the same as the No. 3 seam just below it, but it always has two binders, and usually three. Most of the seam is covered with a layer of what is locally termed "black rash," a series of interstratified thin slates and coals, running to a thickness in places of 4 ft. and averaging probably 16 in. thick. The roof above this is usually sandstone or a hard sandy slate, and roof conditions are generally better than in the No. 3 seam. The black rash is always taken down in headings and usually comes down with, or soon after, the coal in rooms; it increases

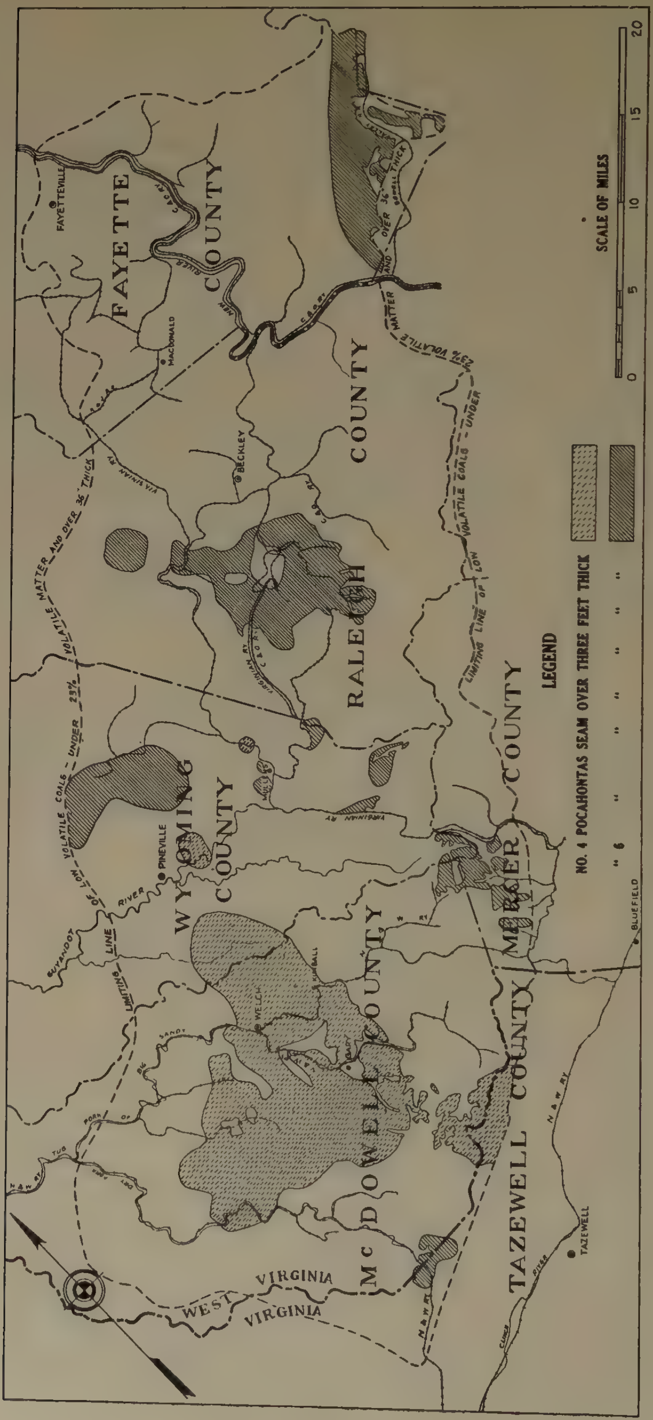

FIG. 4.—SMOKELESS COAL FIELDS OF SOUTHERN WEST VIRGINIA, SHOWING MINABLE AREAS IN NO. 4 AND NO. 6 POCAHONTAS SEAMS.

the ash in the coal, as shipped, considerably. Around Coalwood and between there and Dry Fork this rash is absent, and the seam never has more than two partings and a sandy shale or sandstone roof. It is not so regular as the No. 3 seam.

The McDowell-Wyoming County Report of the West Virginia Geological Survey shows the data about the No. 4 Pocahontas seam, given in Table 2.

TABLE 2.—*Pocahontas Seams No. 4 and No. 5*

County	Number of Measurements	Thickness, Inches								
		Clean Coal			Impurities			Total Seam		
		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
No. 4 Pocahontas Seam										
Tazewell..........	2	62	51	56.5	4	4	4.0	66	55	60.5
McDowell.........	52	95.5	44	63.9	23		5.5	103	46	69.4
Wyoming..........	7	68	35	44.1	9		1.9	68	35	45.9
Total average....	61	95.5	35	61.3	23		5.0	103	35	66.3
No. 5 Pocahontas Seam										
Tazewell..........	19	121	39	63.3	13		2.5	128	39	65.8
McDowell.........	42	87.5	35	47.4	7.5		0.3	95	35	47.7
Total average....	61	121	35	52.4	13		0.9	128	35	53.3

The No. 5 Pocahontas Seam has been found in workable thickness only at the heads of Big Creek and Dry and Beech Forks and on Horsepen Creek and Jacobs Fork in McDowell and Tazewell counties. It has about the same chemical and physical character as the No. 3 seam below it, except that it is slightly higher in volatile matter. The seam is usually clean, but at its greatest thickness has two or more thin partings. The roof is usually a slate or sandstone and mining conditions should be at least equal to the other Pocahontas seams. Its area is shown on Fig. 5.

The Tazewell and McDowell County State Geological Survey reports, and private records, show the data about the No. 5 Pocahontas seam that is given in Table 2.

The No. 6 Pocahontas Seam has been found in workable thickness and has been mined in Fayette, Raleigh, Wyoming, Mercer, McDowell, Summers and Tazewell counties. In Wyoming and Mercer counties it is usually clean and of good quality and has good mining conditions. In Fayette County it has splendid top and usually one heavy parting. The extent of the seam is shown on Fig. 4.

The County Reports of the West Virginia Geological Survey show the data about this seam, where over 3 ft. thick, given in Table 3.

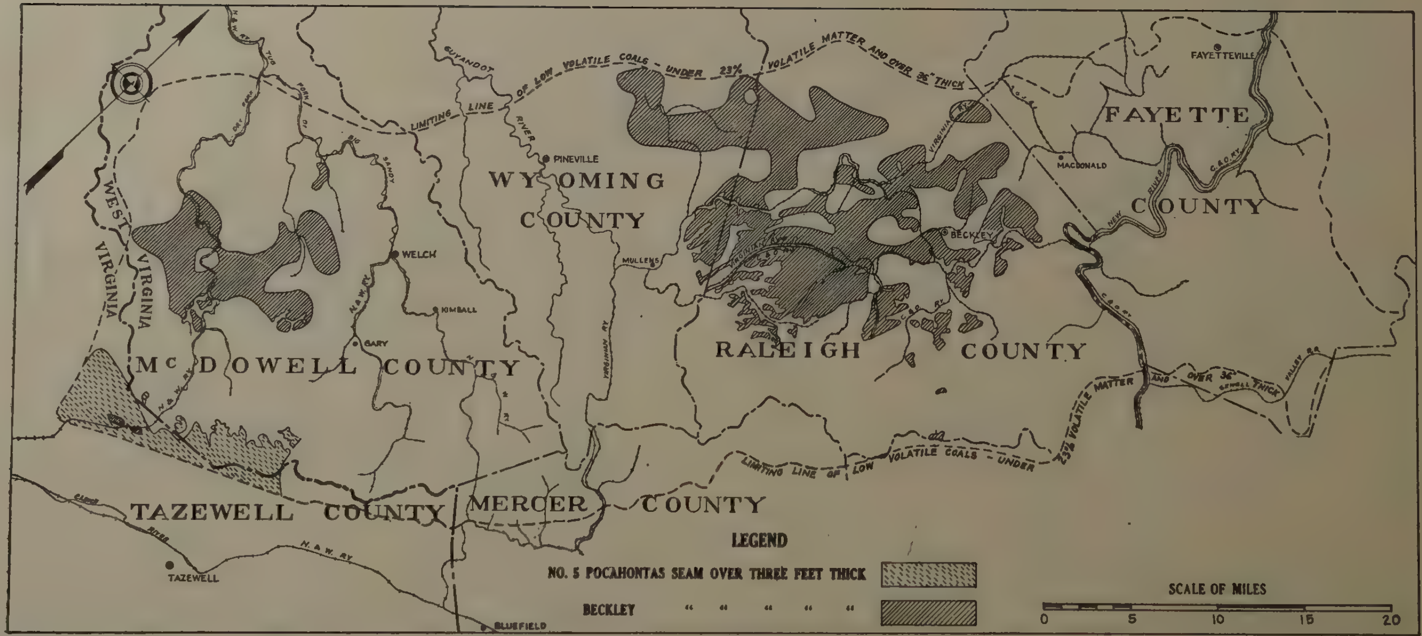

FIG. 5.—SMOKELESS COAL FIELDS OF SOUTHERN WEST VIRGINIA, SHOWING MINABLE AREAS IN NO. 5 POCAHONTAS AND BECKLEY SEAMS.

TABLE 3.—*No. 6 Pocahontas Seam*

County	Number Of Measurements	THICKNESS - INCHES								
		CLEAN COAL			IMPURITIES			TOTAL SEAM		
		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
Fayette	19	72	38	50.2	8	-	2.9	79	38	53.0
Raleigh	10	47	33	39.2	9	-	4.3	52	36	43.5
Wyoming	16	54	33.5	40.7	14	-	2.4	54	36	43.1
Mercer	7	60	35	46.9	15	-	6.6	74	38	53.5
McDowell	2	44	42	43.0	-	-	-	44	42	43.0
Total Average	53	72	33	44.7	15	-	3.4	79	36	48.1

The Fire Creek, or Quinnimont Seam, was first opened along New River, in Fayette County, and has been extensively mined there. Prospecting and drilling has shown its presence in workable thickness in Raleigh and McDowell counties, although it has never been mined in McDowell County. Where it has been operated the seam has been found to be irregular and subject to sudden changes in thickness. It is usually clean, but sometimes, especially where thick, it has a parting or some bad coal at the top which comes down in mining. The roof is usually fair but sometimes has a draw slate which makes a bad top. The grades are irregular and vary as much as the thickness. The quality of the coal is excellent; its irregularity, where mined, shows the need of careful prospecting and drilling. The extent of the Fire Creek seam is shown on Fig. 6.

The County Reports of the West Virginia Geological Survey show the data on this seam, given in Table 4.

The Beckley Seam, or War Creek seam, as it is called in McDowell County, was first opened in Raleigh County and its most extensive development is there. Drilling has shown a considerable area of it in Wyoming County and some large developments have recently been made there. The seam usually has a parting, but frequently is clean, and the roof conditions are usually good, although sometimes a draw slate is found on top of the coal, which makes a bad roof.

The seam is irregular in both thickness and grades and numerous "wants" are found. In Raleigh County it is of excellent quality and in many places its mining conditions make it one of the most desirable of the smokeless coals. In McDowell County the mining conditions are unusually good, but a streak of high ash—10 to 18 per cent—is found in the bottom of the seam which prevents its use for metallurgical purposes. This high-ash portion makes large lumps and when separated brings at least as much as the remainder of the seam for domestic purposes. The extent of the Beckley seam is shown on Fig. 5.

The County Reports of the West Virginia Geological Survey show the data about this seam where over 3 ft. thick which are given in Table 4.

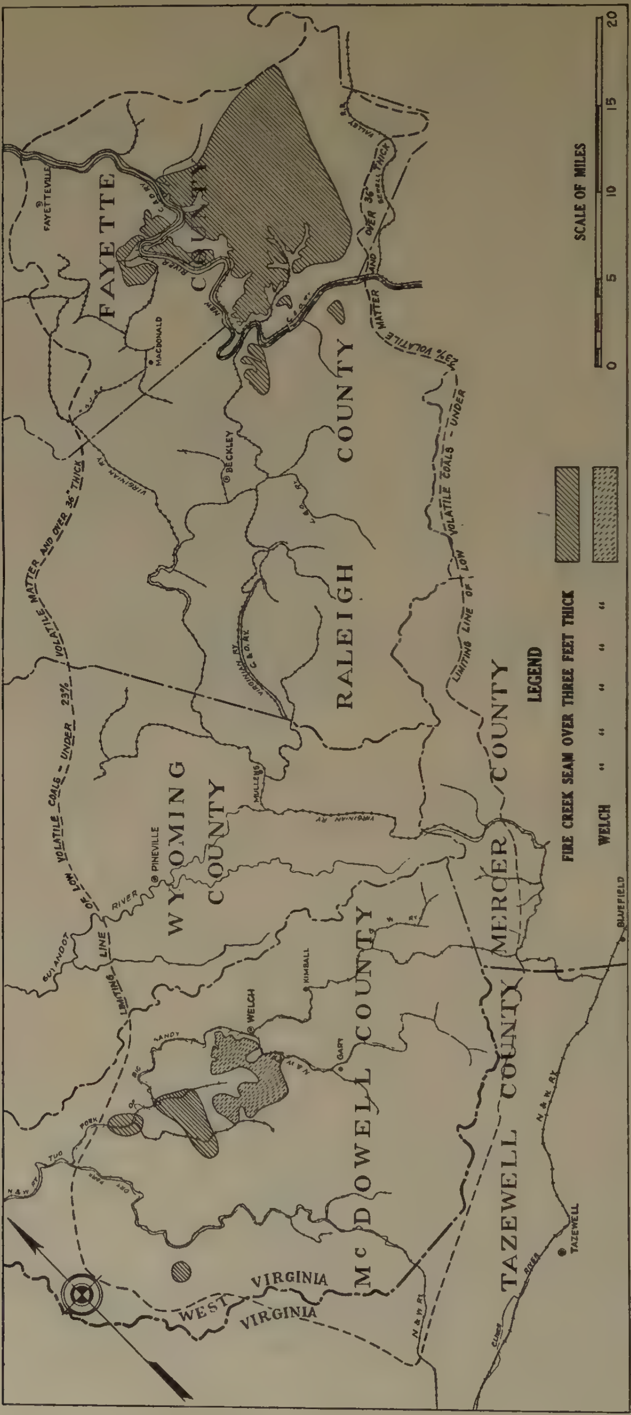

FIG. 6.—SMOKELESS COAL FIELDS OF SOUTHERN WEST VIRGINIA, SHOWING MINABLE AREAS IN FIRE CREEK AND WELCH SEAMS.

The Sewell Seam is the highest one, geologically, that has been worked extensively, and this coal has given the New River field its reputation. The seam is almost always clean and is lower in ash than any other in the series. The top is usually good, but in some places considerable areas have a bad slate top. Grades are fairly regular, but in large areas they are occasionally steep and some "wants" are encountered in mining.

TABLE 4.—*Fire Creek, Beckley, Sewell, Welch and Iaeger Seams*

County	Number of Measurements	Thickness, Inches								
		Clean Coal			Impurities			Total Seam		
		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
Fire Creek Seam										
Fayette	122	84	31.5	48.1	18		0.9	84	32.5	49.0
Raleigh	60	66	34	48.7	19		3.8	84	36	52.5
McDowell	7	62	37	46.6	25		9.6	75	37	56.2
Total average	189	84	31.5	48.1	25		2.1	84	32.5	50.4
Beckley Seam										
Raleigh	184	116	36	62.9	31		4.0	143.5	36	66.9
Wyoming	21	73	34	45.6	28		4.9	99	36	50.6
McDowell	29	96	33	50.7	14		3.5	103	36	54.2
Total average	234	116	33	59.8	31		4.0	143.5	36	63.9
Sewell Seam										
Fayette	225	77	32	54.1	20		1.4	77	32	55.5
Raleigh	80	113[a]	33.5	51.5	17		2.5	126[a]	36	54.0
Wyoming	35	60	33	41.7	22		3.5	67	36	45.2
McDowell	18	48	30.5	39.6	19		3.3	67	32	42.9
Total average	358	113	30.5	51.6	22		1.9	126	32	53.5
Welch Seam										
McDowell	15	57	33	43.0	3		0.5	58		43.5
Iaeger Seam										
McDowell	17	51	23	37.4	16		7	56	36	44.4

[a] Found only in one drill hole, evidently local, as no thickness over 77 in. has been found elsewhere.

Its greatest thickness is in the neighborhood of Macdonald-Glen Jean, and in every direction from that locality it decreases. The thickest coal found in this seam by the West Virginia Geological Survey, with the exception of one drill hole, was 77 in., and some of it is being mined as thin as 30 inches.

In McDowell County, on Tug River, west of Welch, a considerable area of this seam, called locally the "Davy" seam, has been mined for years. While rarely reaching 48 in. in thickness, the seam is clean and the mining conditions are good. The coal is of exceedingly high quality.

The Sewell seam has been opened on Meadow and Greenbrier rivers in mining condition, but here, as well as in a large part of the territory immediately north of the New River, the volatile matter is higher than 23 per cent, thus putting this portion of the field out of the low-volatile class.

The extent of the Sewell seam is shown on Fig. 7. The County Reports of the West Virginia Geological Survey show 358 sections on this seam, with the data given in Table 4.

The Welch, Hemphill or Lambert Seam, as it has been variously called, occurs about 80 ft. below the Sewell or Davy seam, in the territory immediately west of Welch and between that place and Coalwood. It is a local development only and the seam is being worked from 2 ft. 8 in. to 5 ft. 0 in. thick. It usually has a good top and always a streak of "gray" or high-ash coal, which is sometimes picked out but usually shipped. The seam is irregular, both in thickness and grades, and requires careful prospecting before development. Its extent is shown on Fig. 6.

The McDowell County Report of the West Virginia Geological Survey shows openings with the data given in Table 4.

The Iaeger, or Bradshaw, Seam is the highest member of the smokeless series and has had a local development on Dry Fork, in the vicinity of Bradshaw, McDowell County. The seam is usually clean, but occasionally has a parting, and the roof conditions are usually good. The seam is very irregular, both in thickness and grades, and requires careful prospecting before development. Its extent is shown on Fig. 7.

Private reports show the data on this seam, where it is over 3 ft. thick, given in Table 4.

QUALITIES OF SEAMS

The coal from all of these seams is soft and friable and even with the most careful mining and handling a large percentage of slack is produced. Fortunately, most of this can be used for metallurgical and bunker purposes, while the screened coal is in large demand for smokeless domestic fuel. The difference in friability between different seams is usually not much more than the difference between various mines of the same seam.

All of these seams have been extensively sampled and analyzed by the United States and West Virginia Geological Surveys and the analyses in Table 5, except for the Iaeger seam, are compiled from these sources, and include all of those having less than 23 per cent volatile matter. The averages are by counties and for the entire area of the seam, and include all analyses given by both of the authorities mentioned.

TABLE 5.—*Average Proximate Analyses*

Seam	County	Number Of Samples	Moisture	Volatile Matter	Fixed Carbon	Ash	Sulphur	B. T. U.	VOLATILE MATTER		ASH		SULPHUR	
									Maximum	Minimum	Maximum	Minimum	Maximum	Minimum
Iaeger	McDowell	6	-	21.35	72.35	6.29	1.06	14,742	23.80	20.16	8.54	2.96	1.13	0.81
Sewell	Fayette	304	2.63	20.97	72.78	3.25	0.78	14,774	25.45	15.50	8.94	1.74	3.07	0.42
	Raleigh	95	2.80	18.26	74.43	3.44	0.78	14,741	21.87	14.12	5.54	1.54	2.17	0.47
	McDowell	109	2.58	18.31	73.94	5.07	0.64	14,465	20.94	13.50	9.12	2.10	0.98	0.47
	Wyoming	5	0.60	23.31	72.76	3.33	0.86	-	23.77	21.93	4.03	2.34	1.31	0.52
	Average	513	2.63	19.92	73.33	3.67	0.75	14,701						
Welch	McDowell	62	2.28	17.57	72.95	7.58	0.59	14,162	22.39	13.04	9.99	5.41	1.33	0.39
Beckley	Raleigh	164	2.82	16.68	75.56	4.93	0.63	14,476	20.34	13.00	11.10	2.33	1.20	0.47
	McDowell	23	1.77	16.42	72.68	9.13	0.83	13,898	19.15	12.36	11.98	6.91	1.78	0.56
	Wyoming	1	0.29	22.03	75.62	2.06	1.44	-	22.03	22.03	2.06	2.06	1.44	1.44
	Average	188	2.68	16.68	75.21	5.43	0.66	14,404						
Fire Creek	Fayette	140	2.53	17.43	74.76	5.28	0.68	14,641	22.67	14.06	8.57	2.20	1.09	0.41
	Wyoming	1	0.76	21.04	74.98	3.22	0.65	-	21.04	21.04	3.22	3.22	0.65	0.65
	Raleigh	9	1.92	18.00	76.24	3.83	0.67	14,911	23.01	15.99	5.35	1.57	0.76	0.62
	McDowell	1	0.82	21.07	76.13	1.98	0.77	-	21.07	21.07	1.98	1.98	0.77	0.77
	Average	151	2.47	17.51	74.86	5.16	0.68	14,657						
Pocahontas No. 6	Mercer	15	3.35	19.84	73.04	3.83	0.66	14,629	25.22	18.73	5.64	1.93	0.88	0.59
	Fayette	3	0.58	20.89	75.78	2.75	0.54	14,850	21.60	19.90	3.47	2.36	0.66	0.43
	Wyoming	2	0.44	20.32	75.13	4.10	0.60	-	21.29	19.36	4.91	3.29	0.84	0.76
	Raleigh	1	0.80	17.77	78.67	2.76	0.81	-	17.77	17.77	2.76	2.76	0.81	0.81
	Average	21	2.56	19.94	73.90	3.65	0.66	14,665						
Pocahontas No. 5	McDowell	13	0.97	21.49	73.90	4.09	0.53	-	24.24	18.54	6.82	1.91	0.72	0.37
	Tazewell	12	2.31	21.08	72.30	5.01	0.65	-	23.34	18.50	6.81	1.95	0.93	0.49
	Average	25	1.61	21.29	73.13	4.53	0.59							
Pocahontas No. 4	McDowell	50	1.73	15.97	76.27	6.02	0.65	14,619	18.58	13.03	8.80	3.01	0.97	0.40
Pocahontas No. 3	Tazewell	86	2.93	21.12	71.34	4.60	0.54	14,579	21.80	19.60	5.00	3.10	0.59	0.39
	McDowell	437	2.53	16.61	74.94	5.92	0.60	14,362	23.95	11.50	11.38	2.34	1.15	0.41
	Mercer	185	3.01	16.77	75.45	4.76	0.61	14,534	20.29	12.29	15.04	3.14	0.93	0.50
	Wyoming	3	0.61	18.75	75.45	4.86	0.81	-	19.66	17.79	6.35	4.55	0.93	0.65
	Raleigh	6	0.71	18.39	75.97	4.93	0.73	-	19.12	17.16	7.47	3.13	0.85	0.54
	Average	717	2.68	17.22	74.65	5.45	0.60	14,433						

TABLE 6.—*By-product Analyses of Various Seams*

	POCAHONTAS NO. 3			SEWELL			BECKLEY			FIRE CREEK			POCAHONTAS NO. 4 *			POCAHONTAS NO. 5 *		
	Average	Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum
Number Of Samples	6	-	-	11	-	-	9	-	-	2	-	-	2	-	-	6	-	-
Seam Thickness	61"	72"	47"	53"	67"	50"	73"	101"	58"	47"	50"	45"	-	-	-	-	-	-
Sulphur	0.66	0.78	0.55	0.96	1.56	0.50	0.85	1.11	0.58	0.76	0.84	0.67	0.61	0.66	0.55	0.71	0.93	0.52
Phosphorus	0.006	0.014	0.001	0.005	0.012	0.002	0.010	0.039	0.003	0.017	0.018	0.016	0.003	0.004	0.001	0.004	0.007	0.002
Ash	4.40	6.02	3.18	3.05	5.24	1.92	3.96	5.33	2.38	3.18	3.45	2.91	4.67	5.02	4.32	5.67	6.32	5.09
Fixed Carbon	76.59	79.30	73.28	74.02	77.96	68.22	77.59	79.23	75.63	76.37	76.53	76.20	73.39	73.51	73.28	72.92	75.24	71.58
Volatile Matter	19.01	22.40	14.68	22.93	26.54	19.97	18.52	20.06	17.26	20.46	20.89	20.02	21.93	22.40	21.47	21.40	22.40	19.67
Gas Yield - Cubic Feet	8835	9966	7992	10189	10484	9624	9923	10532	9364	10135	10138	10132	8546	8906	8187	9274	9728	8646
B. T. U. Value†	528	613	505	523	553	500	505	539	489	498	502	494	570	613	526	563	617	530
Yield Coke - Per Cent	82.8	86.8	79.1	80.5	82.2	77.8	83.2	84.4	82.0	82.1	82.6	81.7	79.0	79.7	78.4	81.4	81.9	80.4
Yield Ammonia Sulphate - Lb.	18.4	22.2	15.4	23.3	25.0	21.4	21.9	23.8	19.6	20.9	21.0	20.8	20.9	22.2	19.6	19.3	25.0	16.0
Yield Benzol - Lb.	11.5	15.7	9.5	15.6	19.6	10.4	12.3	18.8	7.7	12.0	14.2	9.8	13.9	15.7	12.1	14.7	17.1	12.3
Yield Tar - Gallons	5.4	10.4	3.0	4.9	7.4	3.4	3.8	5.6	2.6	3.7	4.8	2.6	8.9	10.4	7.4	4.5	6.0	3.6

* These samples were not from operating mines and are higher in volatile matter than the average of the seam.
† Per cubic foot.

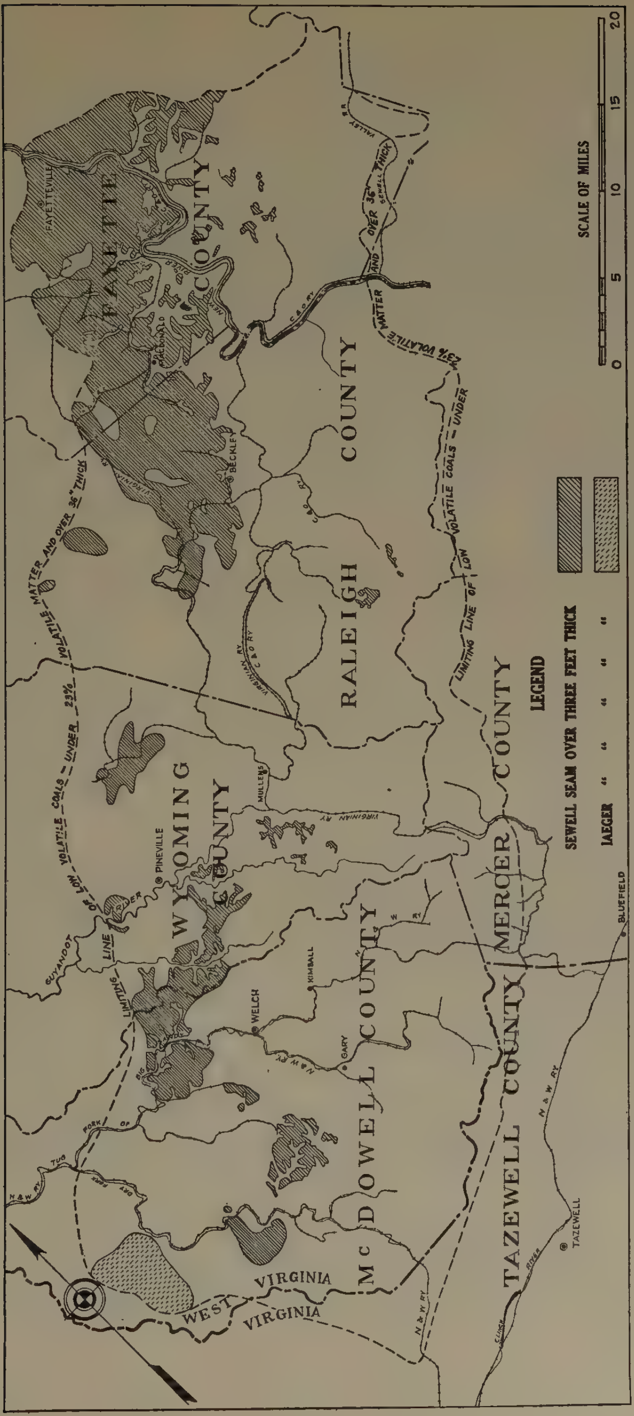

FIG. 7.—Smokeless coal fields of southern West Virginia, showing minable coal areas in Sewell and Iaeger seams.

TABLE 7.—*Amount of Recoverable Coal, 3 Feet Thick and Over, Originally in Each Seam*

	County	Seam Thickness	NO. 3 POCAHONTAS			NO. 4 POCAHONTAS			NO. 5 POCAHONTAS			NO. 6 POCAHONTAS			FIRE CREEK		
			Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons
1	McDowell	3' - 4'	20.550	3.4	104,800,000	25,600	3.5	134,400,000	3,390	3.3	16,800,000	1,600	3.5	8,400,000	5,650	3.3	28,000,000
		4' - 5'	31,680	4.6	218,600,000	21,240	4.4	140,200,000	3,330	4.4	22,000,000	290	4.5	2,000,000	650	4.2	4,100,000
		Over 5'	71,450	6.3	675,200,000	37,530	6.2	349,000,000	290	5.8	2,500,000	-	-	-	-	-	-
		Total	123,680	5.4	998,600,000	84,370	4.9	623,600,000	7,010	3.9	41,300,000	1,890	3.7	10,400,000	6,300	3.4	32,100,000
2	Wyoming	3' - 4'	47,230	3.5	248,000,000	7,630	3.4	38,900,000	-	-	-	14,520	3.5	76,200,000	-	-	-
		4' - 5'	42,640	4.5	287,800,000	3,390	4.5	22,900,000	-	-	-	4,190	4.5	28,300,000	-	-	-
		Over 5'	15,550	5.3	123,600,000	170	5.2	1,300,000	-	-	-	-	-	-	-	-	-
		Total	105,420	4.2	659,400,000	11,190	3.8	63,100,000	-	-	-	18,710	3.7	104,500,000	-	-	-
3	Raleigh	3' - 4'	93,200	3.4	475,800,000	-	-	-	-	-	-	26,460	3.5	138,900,000	1,890	3.3	9,400,000
		4' - 5'	10,680	4.4	70,500,000	-	-	-	-	-	-	3,440	4.4	22,700,000	340	4.3	2,200,000
		Over 5'	5,570	5.5	46,000,000	-	-	-	-	-	-	-	-	-	-	-	-
		Total	109,450	3.6	591,800,000	-	-	-	-	-	-	29,900	3.6	161,600,000	2,230	3.5	11,600,000
4	Fayette	3' - 4'	-	-	-	-	-	-	-	-	-	12,570	3.4	64,100,000	52,170	3.5	273,900,000
		4' - 5'	-	-	-	-	-	-	-	-	-	7,580	4.2	47,800,000	1,150	4.3	7,400,000
		Over 5'	-	-	-	-	-	-	-	-	-	1,150	5.3	9,100,000	1,100	5.3	8,700,000
		Total	-	-	-	-	-	-	-	-	-	21,300	4.5	121,000,000	54,420	3.6	290,000,000
5	Summers	3' - 4'	500	3.3	2,500,000	-	-	-	-	-	-	-	-	-	-	-	-
6	Mercer	3' - 4'	14,580	3.8	83,100,000	-	-	-	-	-	-	2,990	3.6	16,100,000	-	-	-
		4' - 5'	7,400	4.5	50,000,000	-	-	-	-	-	-	-	-	-	-	-	-
		Over 5'	4,940	6.7	49,600,000	-	-	-	-	-	-	-	-	-	-	-	-
		Total	26,920	4.5	182,700,000	-	-	-	-	-	-	2,990	3.6	16,100,000	-	-	-
7	Tazewell	3' - 4'	1,680	3.4	8,500,000	170	3.5	900,000	3,560	3.4	19,200,000	710	3.7	3,900,000	-	-	-
		4' - 5'	230	4.5	1,600,000	340	4.5	2,300,000	3,270	4.5	22,100,000	560	4.2	3,500,000	-	-	-
		Over 5'	10,730	7.9	127,200,000	3,560	6.8	36,300,000	6,770	6.2	63,000,000	60	6.0	500,000	-	-	-
		Total	12,620	7.2	137,300,000	4,070	6.5	39,500,000	13,600	5.1	104,300,000	1,330	4.0	7,900,000	-	-	-
8	Total All Counties	3' - 4'	177,720	3.5	922,200,000	33,400	3.5	174,200,000	6,950	3.4	36,000,000	58,850	3.5	307,600,000	59,710	3.5	311,300,000
		4' - 5'	92,630	4.5	628,500,000	24,970	4.4	165,400,000	6,600	4.4	44,100,000	16,060	4.3	104,300,000	2,140	4.3	13,700,000
		Over 5'	108,240	6.3	1,021,600,000	41,260	6.2	386,600,000	7,060	6.2	65,500,000	1,210	5.3	9,600,000	1,100	5.3	8,700,000
		Total	378,590	4.5	2,572,300,000	99,630	4.9	726,200,000	20,610	4.7	145,600,000	76,120	3.7	421,500,000	62,950	3.5	333,700,000

TABLE 7.—(*Continued*)

	BECKLEY			WELCH			SEWELL			IAEGER			TOTAL ALL SEAMS			Seam Thickness	County
	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons	Area Acres	Avg. Thickness Feet	Net Tons		
1	21,470	3.5	112,700,000	7,920	3.8	45,100,000	17,560	3.2	84,300,000	8,000	3.5	42,000,000	111,740	3.4	576,500,000	3' - 4'	McDowell
	3,840	4.4	25,300,000	-	-	-	340	4.2	2,100,000	-	-	-	61,370	4.5	414,300,000	4' - 5'	
	2,410	5.6	20,200,000	-	-	-	-	-	-	-	-	-	111,680	6.2	1,046,900,000	Over 5'	
	27,720	3.8	158,200,000	7,920	3.8	45,100,000	17,900	3.2	86,400,000	8,000	3.5	42,000,000	284,790	4.8	2,037,700,000	Total	
2	12,400	3.5	65,100,000	-	-	-	15,040	3.3	74,400,000	-	-	-	96,820	3.4	502,600,000	3' - 4'	Wyoming
	2,010	4.5	13,600,000	-	-	-	170	4.1	1,000,000	-	-	-	52,400	4.5	353,600,000	4' - 5'	
	4,360	6.1	39,900,000	-	-	-	-	-	-	-	-	-	20,080	5.5	164,800,000	Over 5'	
	18,770	4.2	118,600,000	-	-	-	15,210	3.3	75,400,000	-	-	-	169,300	4.0	1,021,000,000	Total	
3	39,490	3.5	207,300,000	-	-	-	21,980	3.7	122,000,000	-	-	-	183,020	3.5	952,900,000	3' - 4'	Raleigh
	21,920	4.4	144,700,000	-	-	-	13,490	4.3	87,000,000	-	-	-	49,870	4.4	327,100,000	4' - 5'	
	11,480	5.9	101,600,000	-	-	-	860	5.1	6,600,000	-	-	-	17,910	5.7	154,200,000	Over 5'	
	72,890	4.1	453,600,000	-	-	-	36,330	3.9	215,600,000	-	-	-	250,806	3.8	1,434,200,000	Total	
4	-	-	-	-	-	-	39,600	3.2	190,100,000	-	-	-	104,340	3.4	528,100,000	3' - 4'	Fayette
	-	-	-	-	-	-	30,480	4.6	210,300,000	-	-	-	39,210	4.5	265,500,000	4' - 5'	
	-	-	-	-	-	-	1,200	5.3	9,500,000	-	-	-	3,450	5.3	27,300,000	Over 5'	
	-	-	-	-	-	-	71,280	3.8	409,900,000	-	-	-	147,000	3.7	820,900,000	Total	
5	-	-	-	-	-	-	-	-	-	-	-	-	500	3.3	2,500,000	3' - 4'	Summers
6	-	-	-	-	-	-	-	-	-	-	-	-	17,570	3.8	99,200,000	3' - 4'	Mercer
	-	-	-	-	-	-	-	-	-	-	-	-	7,400	4.5	50,000,000	4' - 5'	
	-	-	-	-	-	-	-	-	-	-	-	-	4,940	6.7	49,600,000	Over 5'	
	-	-	-	-	-	-	-	-	-	-	-	-	29,910	4.4	198,800,000	Total	
7	400	3.5	2,100,000	-	-	-	-	-	-	-	-	-	6,500	3.5	34,600,000	3' - 4'	Tazewell
	-	-	-	-	-	-	-	-	-	-	-	-	4,400	4.5	29,500,000	4' - 5'	
	-	-	-	-	-	-	-	-	-	-	-	-	21,120	7.2	227,000,000	Over 5'	
	400	3.5	2,100,000	-	-	-	-	-	-	-	-	-	32,020	6.0	291,100,000	Total	
8	73,760	3.5	387,200,000	7,920	3.8	45,100,000	94,180	3.3	470,800,000	8,000	3.5	42,000,000	520,490	3.5	2,696,400,000	3' - 4'	Total All Counties
	27,770	4.4	183,600,000	-	-	-	44,480	4.5	300,400,000	-	-	-	214,650	4.5	1,440,000,000	4' - 5'	
	18,250	5.9	161,700,000	-	-	-	2,060	5.2	16,100,000	-	-	-	179,180	6.2	1,669,800,000	Over 5'	
	119,780	4.1	732,500,000	7,920	3.8	45,100,000	140,720	3.7	787,300,000	8,000	3.5	42,000,000	914,320	4.2	5,806,200,000	Total	

BY-PRODUCT ANALYSES

The analyses in Table 6 were obtained by the writer from various sources, in the course of his practice, and while there are not enough of them to cover thoroughly such a large territory, it is believed that they are fairly representative, as no effort was made to select the samples.

AREAS AND TONNAGE OF VARIOUS SEAMS

The figures in Table 7 were obtained by plotting on the county geological maps all of the thicknesses measured at the various outcrop openings and in the mines and found by drilling, in each of the seams, and then drawing the outline of the area in which each seam is 3 ft. thick, or over, of workable coal. The areas were then measured by planimeter and the average thickness was calculated. Assuming a yield of 1500 net tons per acre-foot, which is about an 85 per cent recovery, these quantities gave amounts of recoverable coal originally in each seam, in workable condition, 3 ft. and over in thickness, shown in Table 7.

QUANTITIES MINED OUT

The tonnages mined, as of Jan. 1, 1931, were determined from the production figures of the State Mine Inspector's reports and from the shipment sheets published by the Pocahontas Coal Operators' Association, in the latter data allowance being made for one ton of coke shipped as equivalent to two tons of coal, and an addition of 5 per cent for mine power plants and local uses. No figures are available for production on the C. & O. R. R. from 1873 to 1884, inclusive, and these tonnages are estimated. Where tonnages were given without being distributed among the various seams, the same rate for each seam has been used as was produced when data were available. The distribution among seams and among counties is therefore not exactly accurate, for the reason stated and because coal is sometimes mined in one county and shipped from a tipple located in another county. The production by counties and for seams is shown in Table 8.

The low-volatile coal production as compared with the bituminous coal production of the United States and the total coal production of the world is shown on Fig. 8.

QUANTITIES OF COAL REMAINING

In Table 7 the tonnages given are of recoverable coal; by deducting from these the production, in Table 8, the quantities remaining in each county and in each seam are found. The coal remaining, by counties and seams, as of Jan. 1, 1931, is shown in Table 9.

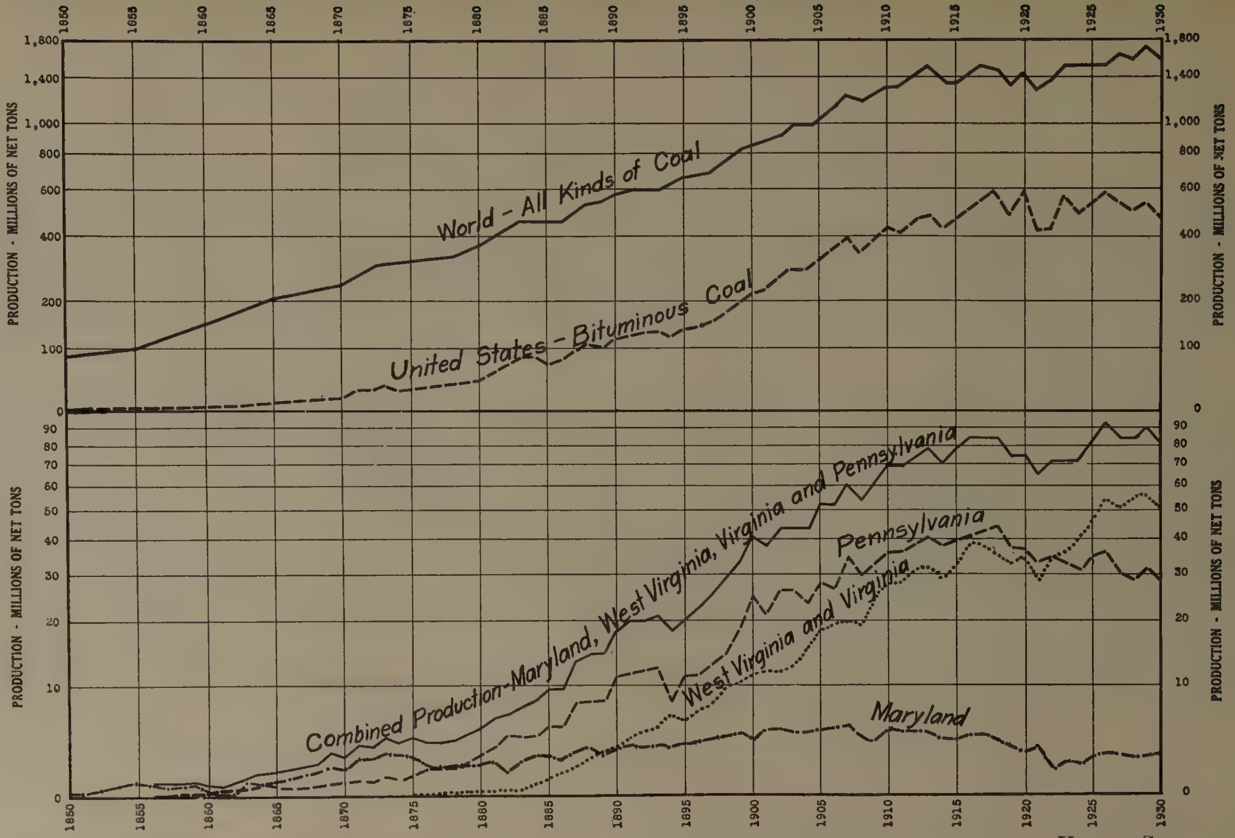

FIG. 8.—UPPER HALF: TOTAL COAL PRODUCTION OF WORLD AND BITUMINOUS COAL PRODUCTION OF UNITED STATES. LOWER HALF: LOW-VOLATILE BITUMINOUS COAL; TOTAL PRODUCTION AND PRODUCTION BY DISTRICTS.

TABLE 8.—*Production by Counties and Seams, to Jan. 1, 1931*

Seam	Fayette	McDowell	Mercer	Raleigh	Tazewell	Wyoming	Totals
Fire Creek	32,635,000			{122,254,000			{190,171,000
Beckley		28,479,000				6,803,000	
Sewell	156,412,000	{48,901,000		37,484,000			{242,797,000
Welch							
Iaeger		1,492,000					1,492,000
Pocahontas No. 6	874,000	2,004,000	970,000	4,460,000	230,000	1,105,000	9,643,000
Pocahontas No. 5		600,000					600,000
Pocahontas No. 4		91,400,000					91,400,000
Pocahontas No. 3		339,880,000	103,576,000	7,793,000	45,192,000	14,904,000	511,345,000
TOTALS	189,921,000	512,756,000	104,546,000	171,991,000	45,422,000	22,812,000	1,047,448,000

Identity of seams uncertain where shown in brackets.

TABLE 9.—*Coal Remaining as of Jan.* 1, 1931

Seam	Summers	Fayette	McDowell	Mercer	Raleigh	Tazewell	Wyoming	Total
Fire Creek		257,365,000	32,100,000		{ 342,946,000			{ 876,029,000
Beckley			129,721,000			2,100,000	111,797,000	
Sewell		253,488,000	{ 82,599,000		178,116,000		75,400,000	{ 589,603,000
Welch								
Iaeger			40,508,000					40,508,000
Pocahontas No. 6		120,126,000	8,396,000	15,130,000	157,140,000	7,670,000	103,395,000	411,857,000
Pocahontas No. 5			40,700,000			104,300,000		145,000,000
Pocahontas No. 4			532,200,000			39,500,000	63,100,000	634,800,000
Pocahontas No. 3	2,500,000		658,720,000	79,124,000	584,007,000	92,108,000	644,496,000	2,060,955,000
TOTALS	2,500,000	630,979,000	1,524,944,000	94,254,000	1,262,209,000	245,678,000	998,188,000	4,758,752,000

Identity of seams uncertain where shown in brackets.

As many of the mines along New River have never drawn any pillars, and as many of those left in place will never be recovered, the tonnages remaining in the Sewell and Fire Creek seams are too large by the amount of this loss. Probably a fair deduction for this loss would be 5,000,000 tons for the Fire Creek seam and 20,000,000 tons for the Sewell seam, all in Fayette County. In the Pocahontas seams the mining practice has been better, and here and in the newer mines in the other counties the recovery of 85 per cent fairly represents actual conditions.

In considering these figures it should be borne in mind that in each of the three fields the best and thickest coal was opened first and most of the mining has been done in this class of coal, so that the reserves are largely of the thinner and poorer qualities of coal from a mining standpoint. In each of these districts the next 15 years will see the completion and abandonment of many of the plants of which the output has made these districts, and the fields, famous for coals of the highest grade for steam, metallurgical and domestic purposes.

DISCUSSION

(*John J. Lincoln presiding*)

C. Scholz, Charleston, W. Va.—No one knows more of the smokeless coal fields of West Virginia than Mr. Eavenson, but I represent some large landowners in the eastern part of Raleigh County on the waters of Glade Creek, where a considerable area of smokeless coal exists which is not shown on the maps accompanying Mr. Eavenson's paper, and I would like to ask why this territory has been overlooked.

H. N. Eavenson.—In the second paragraph of page 74, it is said, "The upper limit of true low-volatile coal is usually considered to be 23 per cent volatile matter; above this point, the coal is usually classed as medium volatile, and this limit will be used in this paper."

There is a great deal of coal in Greenbrier County that will run 25 to 30 per cent volatile matter, and a good deal of it has been shipped. One person from that county said that this paper hurt him because I did not consider that coal as low-volatile. This paper shows where the volatile matter will make the most difference. The upper limit of low-volatile coal is considered by many as 20 per cent instead of 23 per cent. The volatile matter does not interfere with the use of the coal for domestic purposes, but does interfere with it for metallurgical purposes. That is the reason the out-lying sections are not included in this paper.

R. J. Holden, Blacksburg, Va.—There is an increase in the percentage of volatile matter in the coal of the lower seams of this region from about 18 to about 35 per cent for the higher seams, which outcrop to the northwest in West Virginia and to the southwest in Virginia. I am interested in the reason for this difference. Is it due to the greater weight of the sediments formerly overlying this region or is it due to local thrust from the southeast? It might be either one.

H. N. Eavenson.—This paper was written from an engineering point of view and not from a geological point of view. In almost every case the No. 4 Pocahontas seam is lower in volatile matter than the No. 3 seam below it, and a good deal of the Beckley coal is lower in volatile matter than the No. 4 seam below it—certainly not higher. As you go northwest from Pocahontas the volatile matter decreases, running from 21 down to 15 per cent near Welch, and it always seemed to me that this decrease could not be due to the fault in the opposite direction. That is a matter, however, that some geologist will have to pass on.

G. S. Rice, Washington, D. C.—Dr. David White has demonstrated, in his classic studies of the Appalachian field, the great importance of profound horizontal tectonic pressure in changing the composition of coal from a high-volatile stage to a lower volatile stage, and where the natural buttress has given way, as indicated by profound faults parallel with the geologic thrust which have relieved the pressure on that side, the coal is not metamorphosed to the same extent in losing moisture and volatile matter.

Where beds have lain under deep cover there is a greater uniformity of composition. It has been shown in the deep-lying coal beds of Europe that the volatile matter decreases and the fixed carbon increases with depth of the original overlying strata. Also, the effect of heat from volcanic intrusions, as in the Rocky Mountain and Cascade Mountain regions, if not too close to the coal, so as to make natural coke or graphite, lowers the ratio of volatile matter to some degree. To a certain extent it appears that pressure, heat and time in a geologic sense are interchangeable.

P. C. Thomas, Pittsburgh, Pa.—Dr. Rice's statement does not hold for the New River field. The volatile content changes from about 20 per cent in the southern edge of the field to about 25.5 or 26 per cent in the northern edge. The character of the coal changes to a considerable extent. The Sewell coal in the vicinity of Beckley is soft, whereas in the vicinity of Oak Hill it is very hard and firm, making an excellent domestic coal. The same is true in the Beckley seam. The southern end is low volatile. The northern end carries a volatile as high as 21 per cent.

R. F. Roth, Fairmont, W. Va.—Mr. Thomas' remarks on the low-volatile coal fields of southern West Virginia apply also to the Pennsylvania low-volatile fields. In that region there is the real low volatile (under 20 per cent) in a well defined limit—a small area—the northern limit being about Ebensburg, and then south through Cambria County, with the volatile decreasing toward the south and also toward the western limit of Cambria County. The volatile increases rapidly eastward through Cambria County, westward through Indiana County, and northward, northwestward and northeastward through Clearfield, Jefferson, Armstrong, Elk and Center counties. There is a phenomenal situation with regard to the classification of high-volatile and low-volatile coals. The same seam may be classified in part as high-volatile and in part as low-volatile, owing to rapid increases in volatile matter in certain directions, and going north from Ebensburg the volatile matter runs up to 35 per cent while in the south it drops to 16 per cent.

W. E. E. Koepler, Bluefield, W. Va. (written discussion).—The limit of 23 per cent of volatile matter placed on true low-volatile coal, or smokeless coal, by Mr. Eavenson in his paper seems to me to be eminently fair, as it follows the position taken by various governmental and state departments in all of the emergencies during and since the war, when it was necessary, especially, to substitute other coals for anthracite, which was not available in certain territories at these times.

But I wish to present for consideration the viewpoint of people much more interested in this subject than our operators and engineers. The retail coal merchants have the closest contact with this problem because they have to deal with the users of our product and have to face the regulations of the smoke inspectors of the various market cities. Both the consumers and smoke inspectors experience great difficulty in accomplishing a smokeless performance with coals exceeding 22 per cent in volatile matter.

I have been a member of the Smoke Prevention Assn. of the United States for some years, and in July, 1930, I had the privilege of addressing the annual meeting at Newark, N. J. Following my address I was subjected to a long questioning by many smoke inspectors, who are engineers of considerable experience with combustion, and showed a very earnest interest in the subject. They impressed upon me that they felt that the limit of volatile matter in true smokeless coal should be placed not higher than 19 or 20 per cent. While some of our producers of coal do not like this point of view, I think it should be given every consideration because it represents the aspect of actual experience, covering years of hard work and conditions varying from place to place in all sorts of climate and in all sorts of equipment. Every consideration of the limit of 23 per cent as fair and practical is desirable in view of the point of view of the smoke inspectors, retail coal merchants and the public generally.

A. B. Crichton, Johnstown, Pa. (written discussion).—The author excludes a large portion of the New River coal field, located in Greenbrier County, because the volatile content of the Sewell seam in that field slightly exceeds 23 per cent. As a matter of fact, the Greenbrier field has always been considered a part of the New River field, and is thus shown on the maps of the State and United States Geological Surveys, as well as the railroad company maps. The Greenbrier field enjoys the same freight rates, and the coal is shipped in competition with and to the same general markets as the New River and Pocahontas coals.

According to samples of coal taken by various government departments and analyzed by the Bureau of Mines, much of the coal within the lines of the low-volatile, or smokeless, field as indicated on Mr. Eavenson's map contains a higher volatile content than the coals from the Greenbrier field. With the Greenbrier field producing over 3,000,000 tons of coal annually and selling it in competition with the coal from the New River and Pocahontas field, it will be apparent that the conclusions of the author might be considered controversial and unfair to the shippers of coal from the Greenbrier field. As a matter of fact, Bluefield papers in reporting the proceedings did interpret the paper as meaning that the Pocahontas and Winding Gulf fields contained the only true low-volatile or smokeless coals of West Virginia, and coal salesmen in New England and the West have already quoted the paper in an effort to displace Greenbrier coal with Pocahontas coal.

The author might rightly express his opinion as to the minimum volatile content a coal should contain to be considered low-volatile, but when he couples with it the term "smokeless coal" and eliminates the Greenbrier field—with coal running around 25 per cent volatile—he raises a point of controversy which could not but react unfairly to the Greenbrier field. Especially is this so when much of the area included as smokeless contains coals of higher volatile content than the Greenbrier field, and when the Greenbrier coal often runs under 23 per cent.

There has been much controversy anent smokeless coal, not only upon the part of shippers; the courts have wrestled with it, and officials everywhere who are intent upon smoke prevention, with its disastrous results in the congested areas of our large cities, have been trying to define it. It has been suggested that even the lowest volatile coal as ordinarily consumed produces smoke, but it must be admitted that the lower the volatile content, the less smoke the consumer is likely to have. However, "smokeless coal" is a rather indefinite term, for the reason the smokeless burning of coal depends greatly upon other factors as important as the volatile content. The combustion of so-called high-volatile coal in the modern domestic stoker is more efficient and produces less smoke than very low-volatile coal hand-fired.

It might well be claimed by the western Maryland operators that "low-volatile" coal should not exceed 16 to 17 per cent volatile matter; or by the central Pennsylvania operators, 19 to 20 per cent.

It is therefore suggested that the term "low-volatile" or "smokeless" coal is a very general term that has little or no technical significance; that it is a term used in marketing coal, and because of the varying percentages of volatile matter in the different fields has a different meaning in each different market. For the same reason it is suggested that the author in describing the low-volatile or smokeless fields of southern West Virginia should have included the Greenbrier field.

H. N. Eavenson.—As I said in my reply to Mr. Holden, this subject has been considered in this paper entirely from the engineering, not the geological standpoint. This should have been stated more clearly. Many of the data are available in state geological survey reports and in other publications, although not in this form; some are from my own files, and the principal reason for the paper was to place the information where it will be available to the industry at all times.

Some criticism has been made of the use of 23 per cent as the upper limit of low-volatile coal, and my own opinion is that this limit is probably too high. Much of this information was worked up some years ago when the opinions of the limiting factors of low-volatile coals were not as well crystallized as they now are, but time was not available to recalculate all of the figures to a limit of 21 or 20 per cent, which may be decided upon as the upper one. This matter is receiving the careful study of the Coal Classification Committee and will be settled upon the presentation of its final report. If the limit is placed at a lower figure than the one used in this paper, and it almost certainly will be, a class for medium volatile coal will have to be made, and in this will fall a great tonnage of coal not included in this paper, but which still is a most excellent fuel.

The information I had about the Greenbrier field is that little, if any, of it is below the 23 per cent mark, and it was for this reason, of course, that it was not included in this discussion. It is sold as "smokeless" coal, as is much coal running as high as 26 per cent volatile matter, the term "smokeless" being a trade name, of rather elastic quality. It must be remembered that the subject of the paper is the "low-volatile coal field" and that the word "smokeless" is used only to identify it in the public mind.

A question was asked about the amount of captive coal reserves in this area. Approximately they are as follows:

	Net Tons
Totally captive	560,000,000
Partly captive	289,000,000
Total	849,000,000

"Captive" is used to indicate coal that is entirely, or partly, used in plants controlled by its owner.

All of these coals are classified by the United States Geological Survey as semi-bituminous, and have a fuel ratio varying from 8 to 5.

I regret that the data about by-product analyses are not more complete, but they are all that were available.

Wm R. Chedsey.

Some Physical Characteristics of West Virginia Coals

By C. E. Lawall* and C. T. Holland,† Morgantown, W. Va.

(New York Meeting, February, 1932)

When this study was started very little information was available regarding the physical characteristics of West Virginia coals. This was particularly true of friability and of crushing strengths of the coals. Some work had been done on specific gravity studies but the available information applied to only a few coals.

In practically all coal-classification systems thus far proposed, the predominating factors of classifications are largely based on the chemical characteristics of the coal. Recently some investigators have pointed out that a classification based on chemical and physical properties of coal might serve to place coals in their proper group better than classification systems based on chemical properties alone. One of the aims of this study was to obtain some data on the physical properties of coal which might help to classify West Virginia coals.

The Samples

Fig. 1 shows where the coal samples were obtained and the approximate division line of the high and low-volatile coal fields of the state. The relative geological horizons of the coal beds tested are shown in Fig. 2.

In securing samples of coal for friability and specific gravity tests, it was the aim always to obtain a representative sample of lump coal produced at a particular mine. The samples were obtained from the loading boom, the railroad car, or the mine cars. Wherever it was possible to secure samples from a loading boom they were taken from it by taking shovelfuls of coal from the loading boom every minute until about 200 lb. of coal was obtained.

If there were no loading booms used at the mines sampled, the coal sample was taken from a loaded railroad car by systematically taking shovelfuls of coal at definite intervals over the whole top of the car until about 200 lb. of coal was obtained.

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* Director, School of Mines, West Virginia University.

† Research Fellow in Mining, West Virginia University.

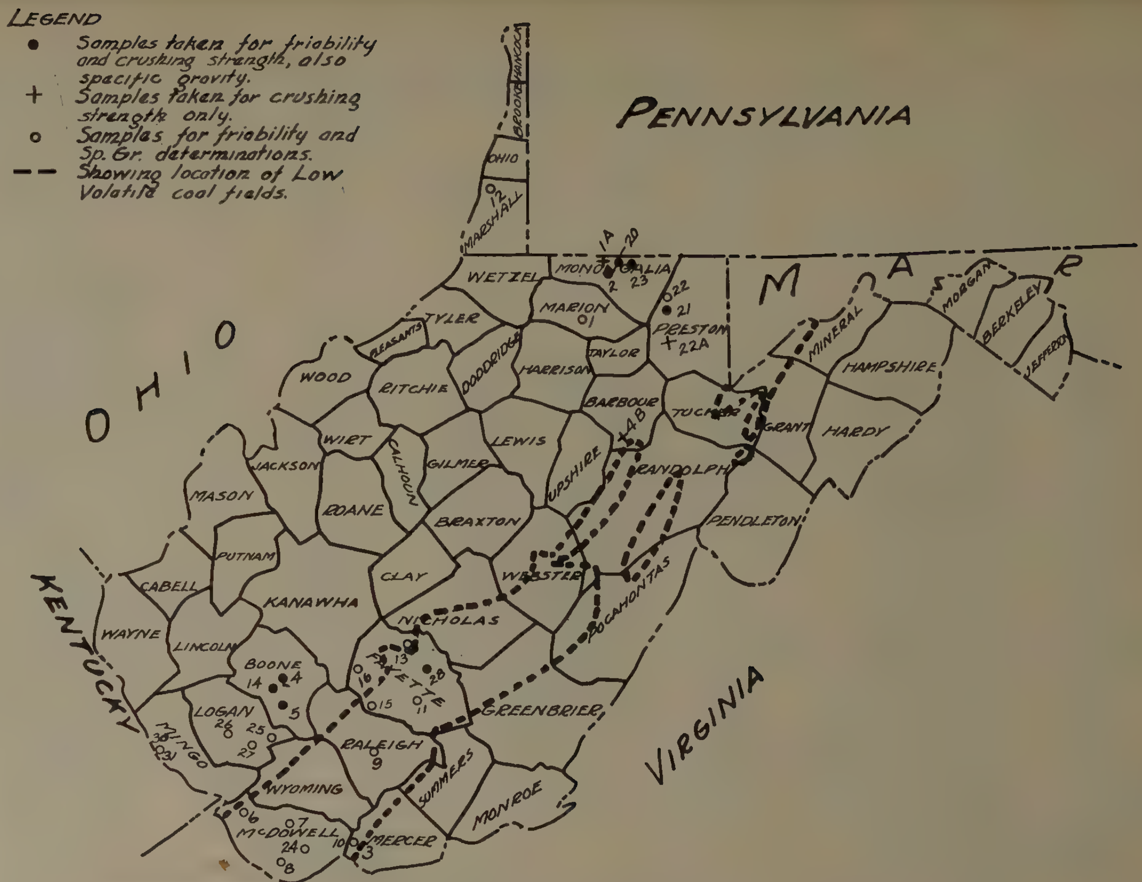

FIG. 1.—MAP OF WEST VIRGINIA SHOWING WHERE COAL SAMPLES WERE OBTAINED AND APPROXIMATE LOCATION OF LOW-VOLATILE COAL FIELDS.

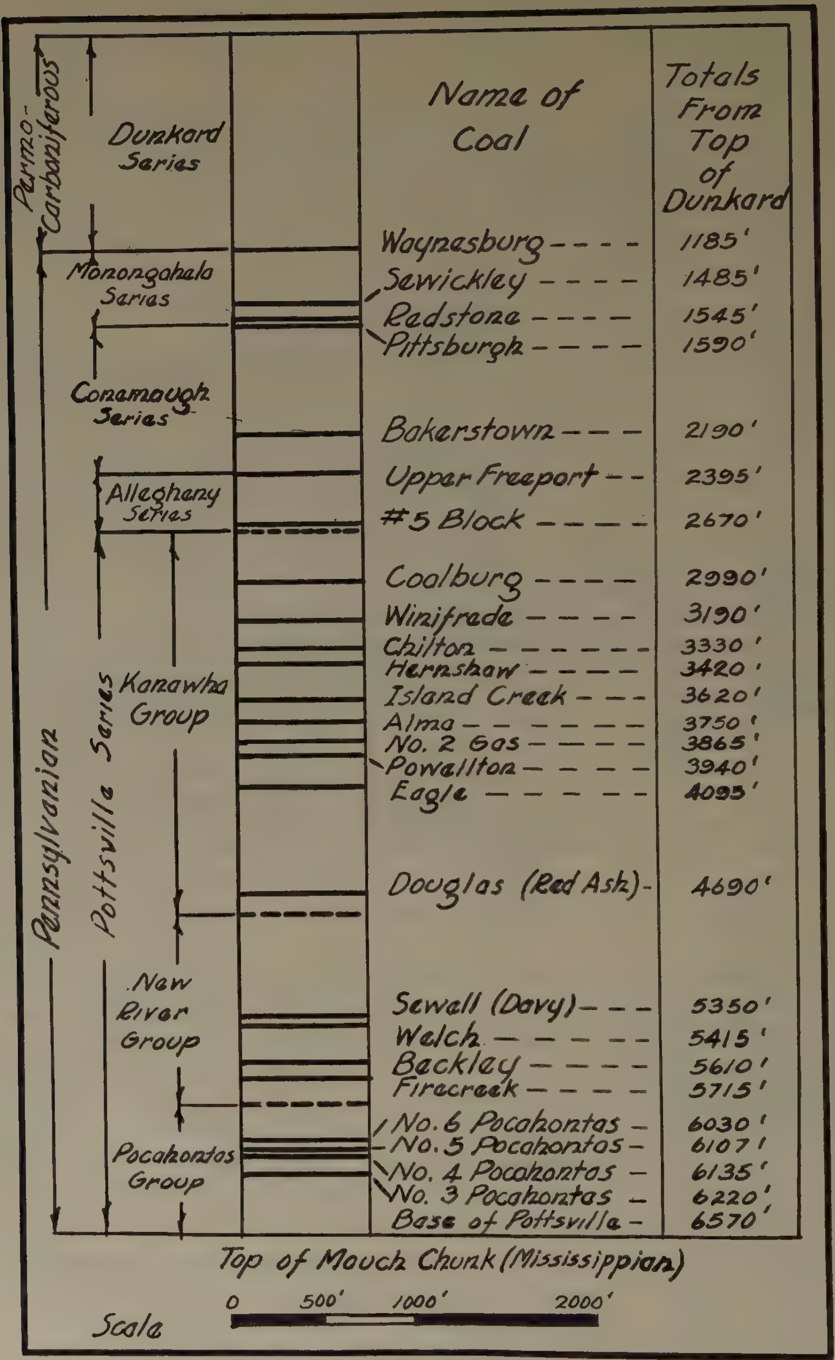

FIG. 2.—GEOLOGICAL HORIZONS OF COALS TESTED.

Wherever coal samples could not properly be obtained from a loading boom or a railroad car, as for instance where coal from two beds was mined and loaded over the same tipple, or where the coal was mined and transported to near-by coke ovens in a larry, the coal sample was obtained from the mine cars taking about 35 lb. of coal from each car until about 200 lb. was obtained.

To obtain coal samples for crushing strengths six or seven lumps of coal constituted a sample. These were taken from loading booms, railroad cars, or mine cars as conditions required. In selecting lumps of coal care was taken to have each lump large enough so that cubes 3 in. on a side could be cut from them. Each lump selected was carefully examined to ascertain that there were no powder cracks or any other visible signs of weakness in it.

FRIABILITIES

The method used for determining the friabilities of the coal samples is practically the same as that used by the United States Bureau of Mines,[1] in some of its work and also the Canada Department of Mines.[2] In general this method consists of putting 1000 grams of coal (approximately 41 pieces) in an Abbe porcelain jar mill. The size of the pieces of coal placed in the jar is determined by selecting pieces that pass through a 1½-in. square-hole screen and remain on a 1.05-in. square-hole screen.

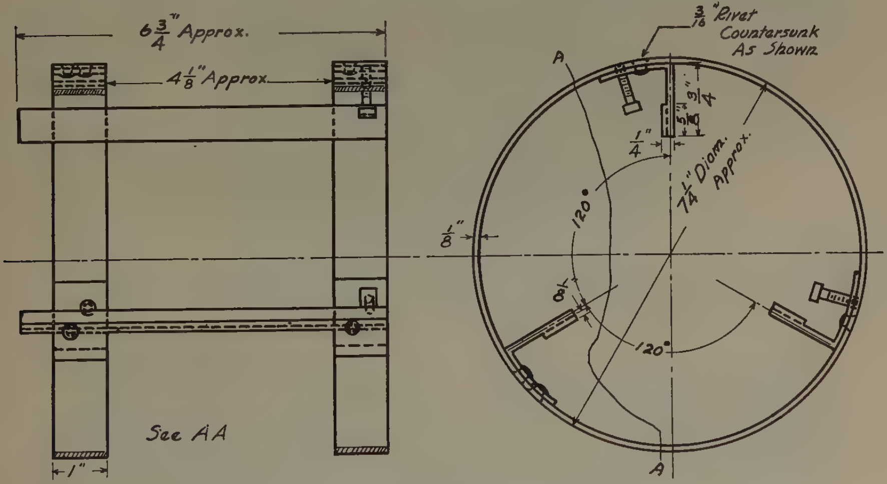

FIG. 3.—DETAILS OF JAR FRAMES WITH LIFTERS.

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[1] Personal communication from H. F. Yancey.

[2] J. H. Nicolls: Friability Tests on Various Solid Fuels Sold in Canada. Canada Dept. of Mines *Bull.* 641.

The inside dimensions of the jar are 7¼-in. dia. by 7¼ in. deep. The jars are fitted with frames and lifters, as shown in Fig. 3 and 4.

After the coal sample is placed in the jar mill, it is rotated 7200 times at 40 r.p.m. At the end of the rotating period the coal is removed from the jar and a screen analysis of it is made. It is screened on standard Tyler screens with openings as follows:

1.50 × 1.05 in.
1.05 × 0.742 in.
0.742 × 0.525 in.
0.525 × 0.371 in.
0.371 × 4 mesh
4 × 8 mesh
8 × 14 mesh
14 × 28 mesh
28 × 48 mesh
48 × 100 mesh
100 × 200 mesh
Through 200 mesh

After screening, the product remaining on each screen is weighed. The pieces of coal remaining on each screen down to the 4-mesh screen are also counted. From these data indices of friability are calculated. Two indices are determined, one based on size of particles and one based on the weight of the particles.

The index of friability is calculated from the following formulas developed by Prof. C. M. Smith, of the University of Illinois:[3]

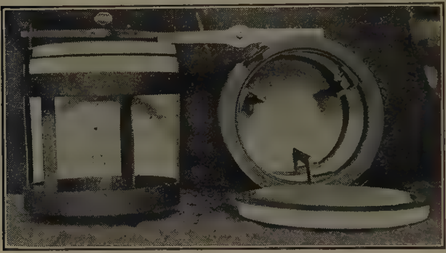

Fig. 4.—Jars and frames.

s = average size of screen openings for various products screened,
w = weight of coal retained on each screen,
S = average size of openings of retaining and passing screens of the sample before rotation.
W = weight of sample of coal tested,
n = number of pieces remaining in each product,
N = numer of pieces in original sample,
D_w = weight degradation,
D_s = degradation in size.

$$D_s = 100\left(1 - \frac{\Sigma ws}{WS}\right)$$

$$D_w = 100\left(1 - \frac{\Sigma(w^2/n)}{W^2/N}\right)$$

CRUSHING STRENGTHS

In order to get some data on the relative strengths of West Virginia coals it was decided to test them in compression. To do this test specimens of regular geometric forms had to be obtained. After experimenting with cylindrical coal specimens cut from solid blocks with a diamond core drill and with cubes sawed from blocks with a hack saw, it was found that the cubes were more satisfactory specimens.

The test specimens were cut from solid pieces of coal by means of a crystolon grinding wheel, 12-in. dia. and 3⁄32 in. thick running at a speed of 2500 r.p.m. The specimens varied from ½ in. thick and 3 by 3 in. in section to 3-in. cubes (Figs. 5 and 6).

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[3] C. M. Smith: An Investigation of the Friability of Different Coals. Univ. of Illinois Eng. Expt. Sta. *Bull.* 196.

FIG. 5.—SOME OF THE COAL SPECIMENS TESTED.

FIG. 6.—THREE-INCH CUBE OF COAL TO BE TESTED IN COMPRESSION.

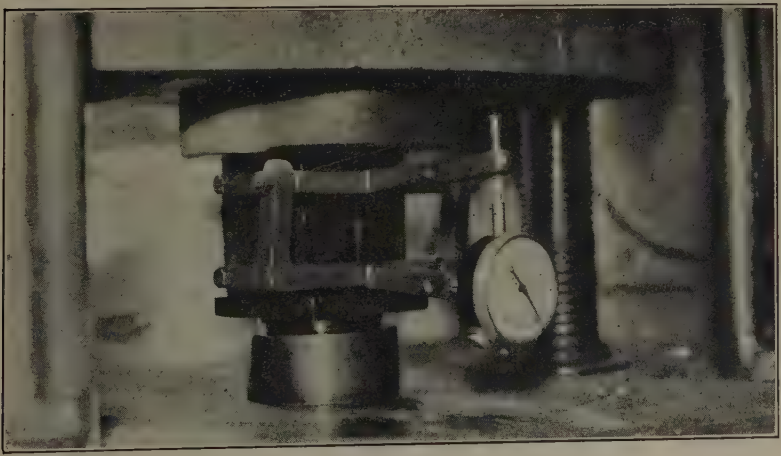

FIG. 7.—STRAIN-GAGE ON A COAL SPECIMEN.

The specimens were tested in compression in a Riehle testing machine by applying the load gradually until the specimens failed. A strain-gage was devised in order to measure the deformation of the specimens under compression as the load was applied (Fig. 7).

Specific Gravities

A portion of each coal sample was taken for specific gravity studies. Each sample weighed approximately 25 lb. (air-dry basis) and was made to pass a 1½-in. screen and remained on a ¼-in. screen. After weighing the sample in air and then immediately immersing it in water and weighing, the specific gravity of the coal was calculated. The sample was left immersed in the water and its weight was recorded each 24 hr. for a period of 10 days. The specific gravity changes of each sample were plotted for this period.

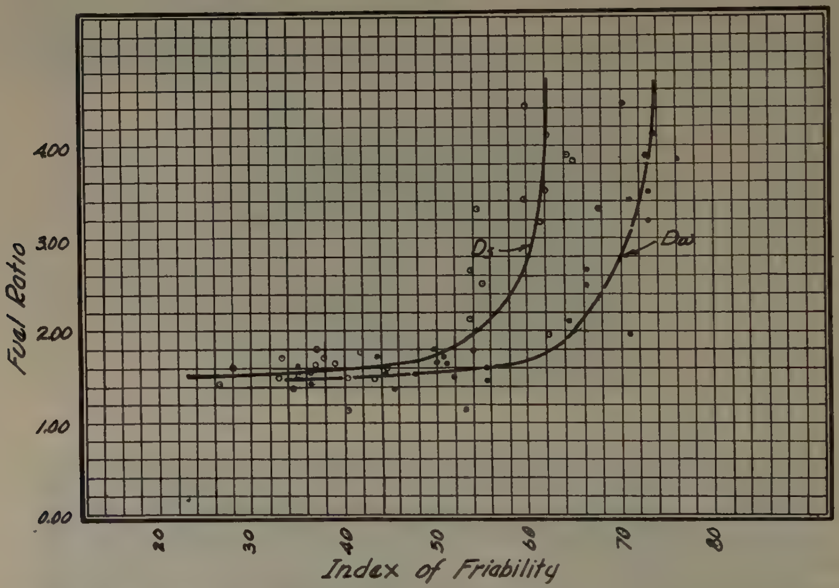

Fig. 8.—Relationship between indices of friability and fuel ratio F.C./V.M.

Relationship between Friability and Fuel Ratio

The curves shown in Fig. 8 are plotted between the fuel ratios F.C./V.M. and the indices of friability of the various coal samples. The coals in these curves arrange themselves so that there is a decided change in the curve at a fuel ratio of approximately 2. The coals represented on the curve above a fuel ratio of approximately 2 include all the New River and Pocahontas coals together with the Bakerstown coal in the Conemaugh series, the Douglas coal in the

TABLE 1.—*Results of Friability Tests of Coals*[a]

Seam	County	Geological Series	Proximate Analysis[b]				Fuel Ratio $\frac{F.C.}{V.M.}$	D_s	D_w	Probable Error ±	
			M	V.M.	F.C.	A				D_s	D_w
1. Pittsburgh	Monongalia	Monongahela	0.8	38.9	53.9	6.4	1.39	34.5	45.8	0.3	0.5
2. Sewickley	Monongalia	Monongahela	1.8	35.4	52.0	10.8	1.47	43.4	55.7	0.6	1.1
3. Pocahontas No. 3	Mercer	Pocahontas	0.6	18.6	72.3	8.5	3.89	64.2	72.7	0.6	1.3
4. No. 5 Block	Boone	Allegheny	2.1	34.6	57.4	5.9	1.66	38.9	50.1	0.9	0.5
5. Hernshaw	Boone	Kanawha	1.5	36.0	58.0	4.5	1.61	27.9	34.9	1.0	1.3
6. Douglas (Red Ash)	McDowell	Kanawha	1.4	26.5	70.3	1.8	2.65	53.7	66.2	0.9	1.5
7. Welch	McDowell	New River	0.8	17.8	73.1	8.3	4.11	62.3	73.7	0.7	0.9
8. Pocahontas No. 5	McDowell	Pocahontas	0.4	20.4	71.6	7.6	3.51	62.2	73.0	0.3	0.6
9. Beckley	Raleigh	New River	0.8	21.7	74.1	3.4	3.41	59.8	70.9	0.9	0.8
10. Pocahontas No. 6	Mercer	Pocahontas	1.0	19.8	76.1	3.1	3.84	64.8	76.1	0.7	1.1
11. Firecreek	Fayette	New River	1.1	23.0	73.3	2.6	3.19	61.6	72.9	0.5	0.8
12. Pittsburgh No. 8	Marshall	Monongahela	1.6	42.8	48.4	7.2	1.13	40.3	53.2	0.8	1.1
13. No. 2 Gas	Fayette	Kanawha	1.3	35.2	60.7	2.8	1.72	33.4	43.7	1.1	1.8
14. Alma	Boone	Kanawha	1.2	36.7	56.4	5.7	1.54	36.4	47.8	0.7	1.3
15. Sewell (Davy)	Fayette	New River	1.5	22.1	73.2	3.2	3.31	54.6	66.8	0.7	0.6
16. Powellton	Fayette	Kanawha	1.3	33.9	60.7	4.1	1.79	41.8	54.0	1.4	2.0
20. Waynesburg	Monongalia	Monongahela	1.7	34.4	51.0	12.9	1.48	40.3	51.9	0.8	1.3
21. Bakerstown	Preston	Conemaugh	1.2	30.2	64.2	4.4	2.13	53.7	64.5	0.6	0.9
22. Freeport, Upper	Monongalia	Allegheny	1.7	30.4	59.5	8.4	1.96	62.2	71.0	0.9	0.3
23. Redstone	Monongalia	Monongahela	1.7	34.1	54.4	9.8	1.60	44.7	55.4	0.8	1.2
24. Pocahontas No. 4	McDowell	Pocahontas	0.5	17.1	75.5	6.9	4.42	60.0	70.1	0.8	1.0
25. Island Creek	Logan	Kanawha	1.4	33.7	60.6	4.3	1.80	37.1	49.7	0.8	1.6
26. Eagle	Logan	Kanawha	1.8	32.6	56.2	9.4	1.72	37.9	50.5	0.6	1.7
27. Chilton	Logan	Kanawha	1.8	35.6	58.5	4.1	1.64	37.1	50.9	0.6	0.8
28. Sewell (Davy)	Fayette	New River	1.4	27.6	68.6	2.4	2.49	55.0	66.2	1.3	1.4
29. Coalburg	Mingo	Kanawha	2.5	33.3	51.6	12.6	1.55	32.9	44.2	1.0	1.8
30. Winifrede	Mingo	Kanawha	2.6	38.5	54.7	4.2	1.42	26.4	36.4	0.5	1.0

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[a] In this table, the higher the degradation number, the more friable the coal.

[b] Proximate analyses used by permission of V. F. Parry and C. J. Potter.

TABLE 2.—*Friabilities of Coals Arranged According to Geological Horizons*

Pennsylvanian								
Monongahela Series			Conemaugh Series			Allegheny Series		
Seam	D_s	D_w	Seam	D_s	D_w	Seam	D_s	D_w
Waynesburg	40.3	51.9	Bakerstown	53.7	64.5	Upper Freeport	62.2	71.0
Sewickley	43.4	55.7				No. 5 Block	38.9	50.1
Redstone	44.7	55.4						
Pittsburgh[a]	40.3	53.2						
Pittsburgh[a]	34.5	45.8						
Average	40.6	52.4	Average	53.7	64.5	Average	50.6	60.6

Pennsylvanian								
Pottsville Series								
Kanawha Group			New River Group			Pocahontas Group		
Seam	D_s	D_w	Seam	D_s	D_w	Seam	D_s	D_w
Coalburg	32.9	44.2	Sewell (Davy[a])	54.6	66.8	No. 6 Pocahontas	64.8	76.1
Winifrede	26.4	36.4	Sewell (Davy[a])	55.0	66.2	No. 5 Pocahontas	62.2	73.0
Chilton	37.1	50.9	Welch	62.3	73.7	No. 4 Pocahontas	60.0	70.1
Hernshaw	27.9	34.9	Beckley	59.8	70.9	No. 3 Pocahontas	64.2	72.7
Island Creek	37.1	49.7	Firecreek	61.6	72.9			
Alma	36.4	47.8						
No. 2 Gas	33.4	43.7						
Powellton	41.8	54.0						
Eagle	37.9	50.5						
Douglas (Red Ash)	53.7	66.2						
Average	36.5	47.8	Average	58.7	70.1	Average	62.8	73.0

[a] Same seam different localities.

Kanawha series and the Upper Freeport coal in the Allegheny series. The coals represented on the curve below a fuel ratio of 2 are coals from the Monongahela, Allegheny, and Kanawha series. From these curves it may be possible to estimate the friability index of a coal if its fuel ratio is known.

If points are plotted on a chart showing the relationship between the fixed carbon and the index of friability (D_s) the coals arrange themselves into two groups, as shown in Fig. 9.

CRUSHING STRENGTHS

It was decided to determine the crushing strength of the coal samples with the load applied perpendicular to the bedding planes and also with the load applied parallel to the bedding planes.

By using a strain-gage on the test pieces of coal while the load was being applied it was possible to determine the modulus of elasticity of

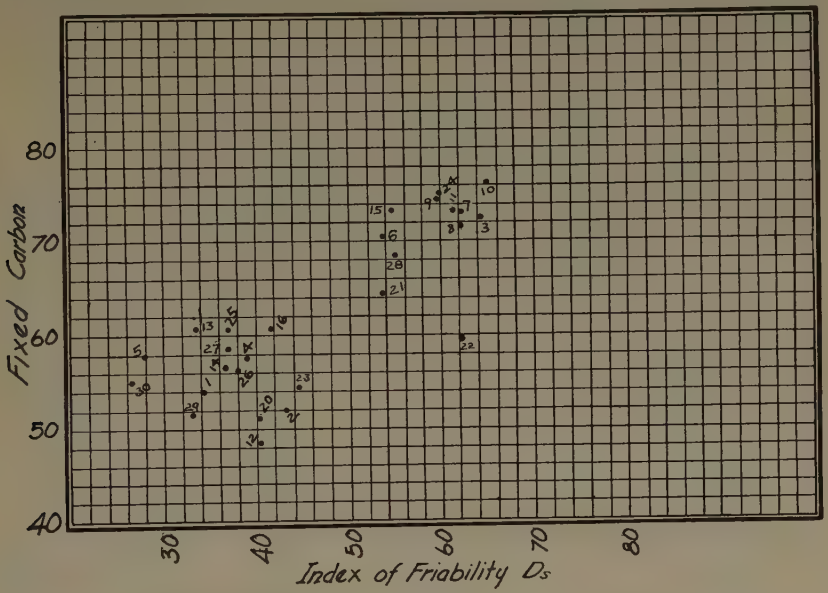

FIG. 9.—RELATIONSHIP BETWEEN FRIABILITY (D_s) AND FIXED CARBON.

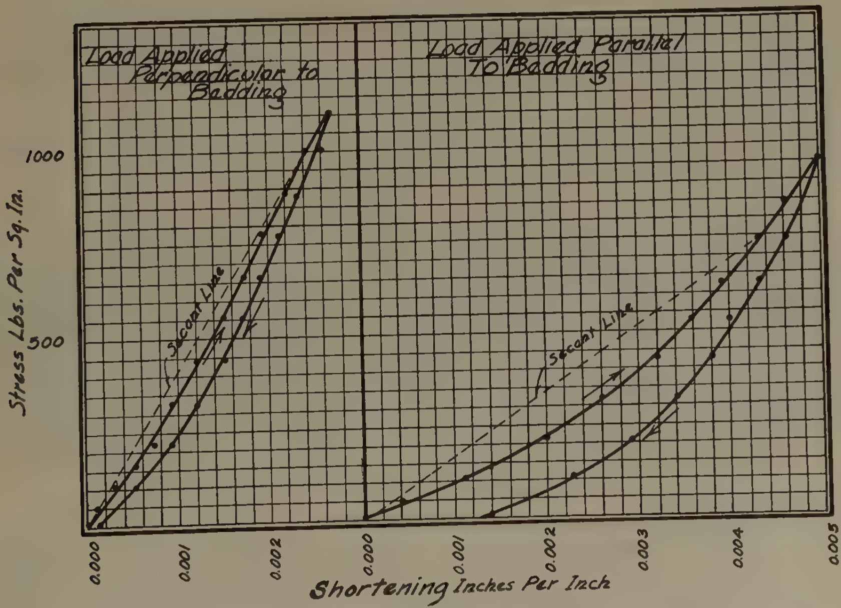

FIG. 10.—TYPICAL STRESS-STRAIN CURVES OF A COAL TESTED.

the coals, with the load applied perpendicular and parallel to the bedding planes. Inasmuch as the stress-strain curves obtained were curved lines, it was decided to determine the secant modulus of elasticity. The secant was drawn between the origin of the curve and the deformation obtained at a stress equal to one-half the ultimate strength of each piece tested (Fig. 10.) The secant modulus was used because the relative stiffnesses of the various coals could be obtained from that curve as well as, or better than, by using any other modulus.

TABLE 3.—*Compression Characteristics*

Size of Sample, 3-in. Cube

Sample No.	Name of Coal	Number of Cubes Tested		Crushing Strength, Lb. per Sq. In.		Secant Modulus of Elasticity	
		Perpendicular to Bedding	Parallel to Bedding	Perpendicular to Bedding	Parallel to Bedding	Perpendicular to Bedding	Parallel to Bedding
21	Bakerstown	12	8	2,420	1,655	550,000	167,500
22	Freeport, Upper	8	6	1,682	1,554	373,000	268,000
28	Sewell	9	9	2,190	1,658	370,000	178,000
1A	Pittsburgh	7	7	2,710	2,400	470,000	385,000
20	Waynesburg (Upper bench)	15	12	4,474	3,244	475,000	800,000
2	Sewickley	12	12	3,630	2,500	615,000	573,000
23	Redstone	10	9	2,932	2,579		
4A	No. 5 Block[a]	11	11	1,819	1,517		
4	No. 5 Block[a]	10	10	4,893	4,208	573,000	593,000
14	Alma	12	10	3,860	3,667	485,000	614,000
5	Hernshaw	10	11	4,727	3,590	550,000	635,000

[a] Same seam, different localities.

Table 3 gives the values of the crushing strengths and the secant moduli of elasticity for various coals tested.

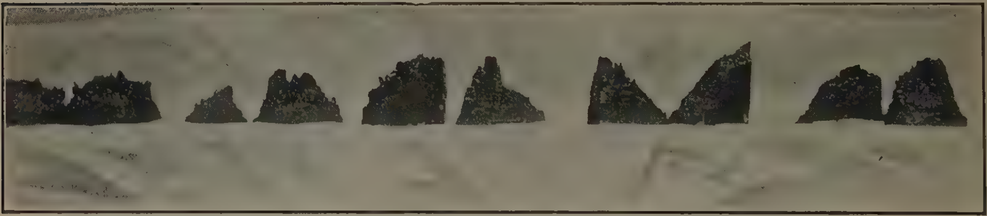

FIG. 11.—TYPICAL COMPRESSION FAILURES OF BITUMINOUS COALS.

When the coal test pieces failed in compression, the type of failure was usually a wedge (Fig. 11), the angles of which varied between wide limits.

In all the compression test work it was realized that the stress that a coal is able to withstand without failing is dependent upon the dimensions of the test piece. By using cubical test pieces it has been found that the larger the dimensions of the cube the smaller will be the unit stress that the coal will withstand.[4] As the ratio between the cross-sectional dimensions and the thickness of the test piece increases the unit stress also increases. In this study the range of sizes of test cubes of coal was not varied a great deal, primarily on account of the difficulty of cutting out

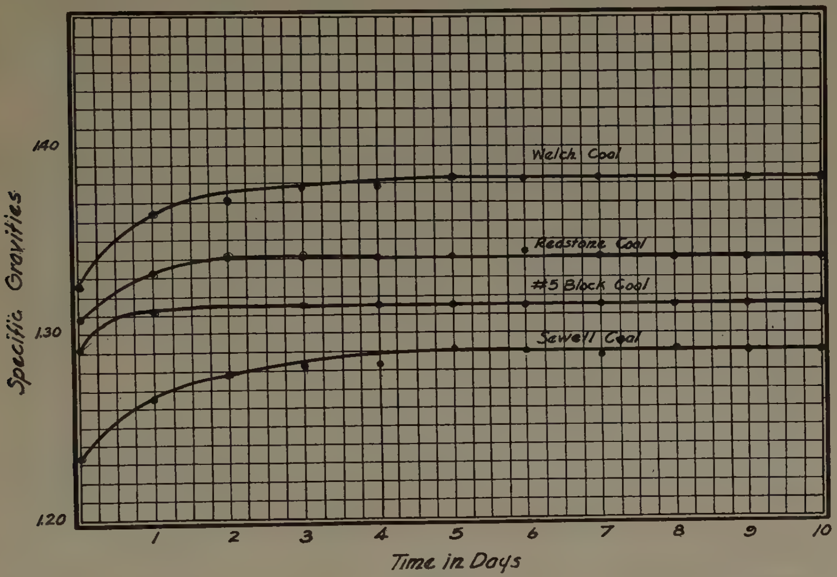

FIG. 12.—INCREASE OF SPECIFIC GRAVITIES OF TYPICAL COALS WHEN IMMERSED IN WATER.

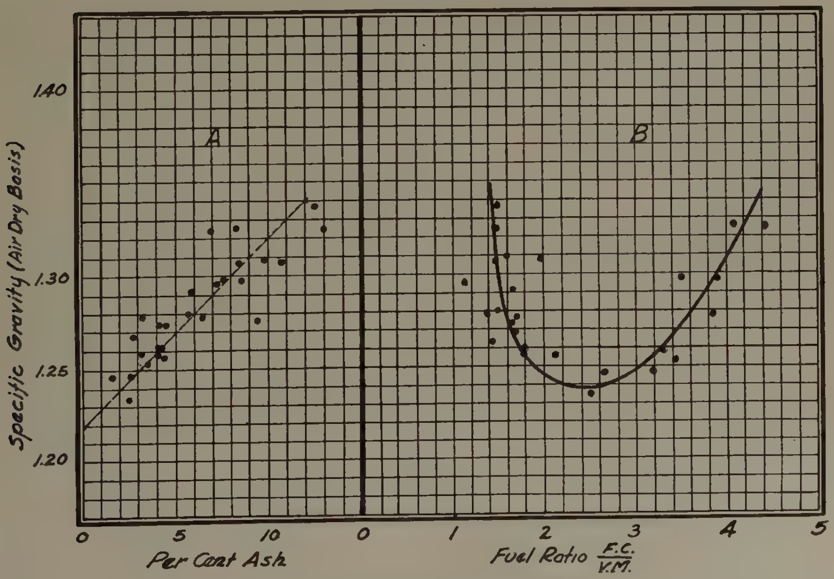

FIG. 13.—ASH AND FUEL RATIO F.C./V.M. OF COALS TESTED WHEN PLOTTED AGAINST SPECIFIC GRAVITIES.

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[4] U. S. Bur. Mines *Bull.* 303.

large cubes of coal from the samples obtained. On one sample, however, 2- and 3-in. cubes were tested and it was found that the 3-in. cubes did not withstand as large a unit stress as the 2-in. cubes. For example, 2-in. cubes of Pittsburgh coal failed at an average unit stress of 4029 lb. per sq. in. (perpendicular to the bedding planes) and 3-in. cubes failed at a unit stress of 2710 lb. per square inch.

Approximately 200 specimens of coal were tested having the ratio of the least cross-sectional dimension to the thickness varied from about 1:1 to 1:10. The stress that the coal was able to withstand became much greater as the ratio of the least sectional dimension to the thickness increased. From the amount of test work completed up to the present time, however, it is impossible to state just how much the ultimate strength varies with this ratio.

TABLE 4.—*Specific Gravities*

No.	Name of Seam	Weight of Sample, Lb.	Specific Gravity Air-dry (Basis)	Specific Gravity after 10 Days Immersed in Water	Percentage Increases
28	Sewell	20.20	1.234	1.290	4.54
6	Douglas (Red Ash)	15.75	1.246	1.300	4.33
11	Firecreek	20.57	1.247	1.299	4.17
9	Beckley	19.84	1.253	1.310	4.55
21	Bakerstown	18.89	1.256	1.308	4.14
16	Powellton	20.13	1.257	1.295	3.02
15	Sewell	23.51	1.259	1.318	4.69
25	Island Creek	26.50	1.261	1.300	3.09
30	Winifrede	19.68	1.262	1.298	2.85
13	No. 2 Gas	21.58	1.268	1.291	1.81
27	Chilton	19.54	1.272	1.303	2.44
5	Hernshaw	20.13	1.273	1.302	2.28
26	Eagle	28.64	1.277	1.310	2.58
1	Pittsburgh	21.59	1.278	1.312	2.66
10	Pocahontas No. 6	19.22	1.278	1.338	4.69
14	Alma	21.51	1.280	1.311	2.42
4	No. 5 Block	22.82	1.291	1.315	1.86
12	Pittsburgh No. 8	31.05	1.295	1.327	2.47
3	Pocahontas No. 3	22.35	1.297	1.364	5.17
8	Pocahontas No. 5	19.44	1.298	1.366	5.24
2	Sewickley	24.51	1.307	1.341	2.60
22	Freeport, Upper	22.33	1.307	1.339	2.45
23	Redstone	22.89	1.309	1.340	2.37
24	Pocahontas No. 4	18.98	1.324	1.364	3.02
20	Waynesburg	20.37	1.324	1.374	3.78
7	Welch	20.13	1.326	1.383	4.30
29	Coalburg	20.10	1.336	1.380	3.29

SPECIFIC GRAVITIES

Results of the specific gravity determinations are given in Table 4. Fig. 12 shows the increase in specific gravity after the coal is immersed in water.

The graphs in Fig. 13 show the relationships between the specific gravities of the coals and percentage of ash (graph *A*) and the relationships between specific gravities and fuel ratio (graph *B*).

From graph *A* it is seen that in general the specific gravity of a coal increases as the ash content increases. The trend is very general and it is very doubtful whether the ash content of a coal could be predicted from its specific gravity.

For the coals studied there is an approximate increase of 0.01 in specific gravity for each one per cent increase in ash, as indicated by the dotted line in Fig. 13*A*. A consideration of the known variations in the chemical composition of coal substance and of ash indicates that this relationship offers only an approximate rule by which to calculate the specific gravity of the pure coal substance or to estimate the percentage of ash in the coals.

Graph *B* shows, for the coals studied, that there is a correlation between the specific gravity of the coals and their fuel ratios (fixed carbon divided by the volatile matter). The coals that arrange themselves on the right limb of the curve above a fuel ratio of 2.5 are the New River and Pocahontas coals and the Douglas from the Kanawha coals. The coals on the left limb of the curve below a fuel ratio of 2.5 are from the Monongahela, Conemaugh, Allegheny and Kanawha measures.

CONCLUSIONS

The data presented in this paper represent only part of the studies that are being carried on regarding the physical characteristics of West Virginia coals. There is no doubt that the chemical studies of coal give very important information for coal classification work but much valuable information can be obtained from the physical properties of coal which may be of aid in classification studies.

The shape of the graphs showing the relationships between friabilities and fuel ratios for the coals studied seems to indicate a boundary between the bituminous and semibituminous coals of West Virginia. The graphs showing the relationships between specific gravities and fuel ratios also seem to indicate a boundary between the West Virginia bituminous and semibituminous coals. It is realized that these data have been obtained from a very few samples of coals but the two graphs are interesting because the boundaries between the two classes of coals are approximately the same.

DISCUSSION

(A. C. Fieldner presiding)

A. C. Fieldner, Washington, D. C.—Aside from coal classification, this information on strength and friability is of considerable importance in connection with mining methods, preparation and transportation. We are greatly indebted to the University of West Virginia for carrying on this investigation.

J. R. Campbell, Pittsburgh, Pa.—From a practical standpoint we are interested in seeing ball mills on a larger scale. We feel that by using a steel lining, instead of a porcelain lining, we would be making a much better machine. It would take a larger sample, and larger size of coal. The coke industry and the big steel companies are evaluating their coke by such a procedure. They are coming to believe that this index is a true measure of the coke for blast-furnace purposes.

From a commercial standpoint, we are all interested in how the coal will prepare in its sizes. This is especially true of coals that are being shipped for domestic use and in domestic sizes. We all know that the anthracite people have a standard for domestic sizes and for steam sizes. I do not know just what the break is at the present, but when I was associated with the industry it was about 9⁄16 in.; the prepared sizes above that size and the steam sizes below. The same general rule would apply in the preparation of bituminous coals with a break at, or near, ½ inch.

In the Pocahontas field we know that the break is at, or near, ½ in., plus for domestic sizes and minus for steam sizes. There is a tendency all the time to get a large range in the top sizes above ½ inch.

In the matter of the mechanical preparation of coals this friability index is very valuable, in that it would show the degradation of the coals into the finer sizes, which is both objectionable to the wet method and the dry method of preparation. In the wet method the 48-mesh by zero material is very difficult to handle in water, and in the dry method it presents a real dust-collection problem.

If these tests could be conducted in a larger machine, a standard machine that could be adopted by all of us who are interested in the preparation of coal, and if that machine would show these indices in the percentage of weights by size, at or near ½ in., or ¼ in., and then again at 48-mesh by zero, it would give those interested in the preparation of coal a very valuable machine.

Naturally, I am coming to look at all of these tests more or less from a practical standpoint. The nearer that we can come to the practical aspects of the coal man's problem, the more service we can give in work of this character.

In discussing classification of coal, it has been hinted that if this could be done on a moisture basis it would be a very simple solution of the coal classification problem. That classification and this classification of a friability index are two very valuable classifications. We have inquiries from people in the West who have high-moisture coals with low B.t.u. They want to know whether there is some means of reducing the moisture in the western coals from a shipping standpoint and from a firing standpoint. Some practical means *would* be valuable.

In addition to these two methods of course we must not lose sight of the rank of the coals, which is probably determined by some other means; I refer to the value of coals for coking purposes. That is distinctly aside from coal that is used generally for steam purposes. We know the rank of the coal has a very important bearing on its value for metallurgical purposes.

G. H. Ashley, Harrisburg, Pa.—The time element, involving probably more or less oxidation of the coal, must be taken into account in sampling high-volatile coals. Great differences in friability can be seen between coal freshly mined in the rooms from that taken from an old pillar, coal on the pillar being much the more friable. Such differences in friability might be indicative of chemical differences that would affect the classification of the coal.

J. R. Campbell.—From a practical standpoint, I believe these tests should be conducted in a tumbler type mill, as being more clearly indicative of what would happen to coal traveling through a tipple on conveyors and dropping here and there. That would be a better test and a more severe test and would show how coal would be handled. I refer to the ball mill type. I believe it will replace the shatter test.

H. N. Eavenson, Pittsburgh, Pa.—Dr. Lawall is to be congratulated on having started this work. We should have a great deal more of it, as it is of great value from a practical standpoint. Several years ago I had occasion to do some work of this kind for a mine in Harlan County, Kentucky, mining the High Splint seam of coal. This is an extremely hard coal and mines in large lumps. A complaint from a customer stated that the coal was not as hard as it had been from that seam, and the reason given was that it was wetter and of a red color. This did not sound like a correct reason, so we had some tests made at the Koppers company laboratory to determine the relative hardness of the black and the red coal. These tests showed that apparently there was no connection between the two. Later, when some of the headings approached the outcrop and the coal became red, the complaints recurred. As the spread between the price received for lump and for the smaller sizes was more than one dollar per ton, it became a real practical problem, as the customers were perfectly willing to pay the high price if the coal was as hard as they thought it should be.

Finally we built a standard coke shatter test machine, using a drop of 6 ft. on to an iron plate, and up to date we have tested 50 samples in this machine, using four drops for each. We find that the amount of breakage is very irregular, but that there is not necessarily any difference between the coal that is weathered and the coal that is black. We have found, however, that there is a great deal of difference in the coal in the center of the mountain and the coal approaching the outcrop, as apparently from 300 to 400 ft. from the outcrop the coal becomes softer, or perhaps somewhat shattered, allowing the larger lumps to break more easily, although there is no difference in the hardness of the pieces when they have been broken to the sizes of egg coal.

The tests were made in a manner comparable to that used in Illinois, as at that time the latter were the only shatter tests available. Lump coal only, and nothing less than at least 8 by 10 in., or the coal that was shipped as block, was used. We intend to continue testing on the headings approaching the outcrop, so that we can tell when the coal may be expected to change, as there is no visual way of telling any difference.

It is desirable that some standard test for the larger sizes of coal of that particular type should be devised and used, as coal of that kind is shipped to the lakes and to other places where degradation is an important matter, and the hardness of the coal is the largest single commercial factor. Several coals are produced in that section that are chemically about the same but vary greatly in hardness. If some standard test can be worked out that will cover these conditions, it will be a valuable addition to the information that we already have.

T. G. Fear, Fairmont, W. Va.—I would like to comment on the practical application of Professor Lawall's paper. By practical I mean something that coal people can take out to the mines and use either to lower the mine costs or increase the realization. About two years ago I worked with Professor Lawall on compressive tests of coal in the Pittsburgh seam in northern West Virginia, which have been quite beneficial. We were having considerable complaint from some of our customers and our sales department was specifying certain mines on many orders. The samples taken were from mines that were approximately 15 miles apart. The tests on these samples at the University showed a considerable variation in the compressive strength of the coal from the same seam but different locations in the district.

I believe that we are going to use this method of finding the strength of our coals to guide us in our shooting. At one mine in the State of West Virginia, where the Joy coal saw was being used, it was learned that three 6-ft. cuts had to be taken out before solid coal which had not been affected by the shattering effects of the powder could be reached. I believe that Professor Lawall's tests are going to be a great help to the coal industry.

C. M. SMITH, Urbana, Ill.—We have adhered to the drop test as a means of determining the friability of coal, feeling that to drop a single layer of coal, only once, we are subjecting it to a very simple and explicit test, whereas in a tumble barrel, or in repeated shatter tests (like those used on coke), the material is being reworked. After it has started to break down we do not know what state the sample is in and hence do not know what we are working on. For that reason, we have adhered to the idea of dropping a single layer of coal only once.

Our scheme is to take a considerable amount of 3 by 2½-in. coal, set it in a single layer on trap doors, spring the doors and drop the coal 10 ft. to concrete. Then we measure the relative amount it has gone down, both in size and average weight per piece, and express it, as Dr. Lawall has shown, in the form of friability or degradation numbers.

I am glad to see the operating men interested in this. We have long had the feeling that the degradation of coal has been overlooked, both as a matter of commercial interest and as a scientific study. However, it is a mean phenomenon to work with. The stuff breaks one way at one time and another way at another time, with the result that a large amount of sampling and testing is necessary to get consistent results. I suppose the reason that nothing more has been done with it is the fact that the phenomenon is so variable and requires so much tedious work.

I have the feeling that among these various schemes of testing—the coke shatter test, barrel test or our drop test, we are going to work out a standard and acceptable test that will give a good clew to the real friability of coal. Certainly we are justified in going ahead toward that end, and anything that the Coal Division can do toward stimulating the work should be well rewarded in the end.

C. E. LAWALL.—To some the "modulus of elasticity" of coal may seem to be a very theoretical term. Still, I believe it has a practical interest to mining men in determining the stress that occurs in the pillars of mines as coal is mined. If, in a mine where the roof and floor are good and not likely to heave under pressure, the height of the coal bed is measured before much coal is mined, and then measured again when the pillars have been deformed by the increase of weight coming on the coal due to mining, it would be possible to determine the unit stress in them if the modulus of elasticity of the coal were determined by the method outlined in my paper. A method would have to be devised, of course, to measure the unit deformation in the pillars, but this would not be a difficult matter. The stress in the pillars could be found even where there is some heaving of the floor and roof if points were fixed directly on the pillars of coal and deformations measured between them.

Classification of the Coals of the Arkansas-Oklahoma Field*

By Thomas A. Hendricks,† Washington, D. C.

(New York Meeting, February, 1932)

The object of this paper is to give a brief description of the coals in the different districts of Arkansas and Oklahoma, their present commercial classification, and the need for a scientific classification that shall be more nearly in accord with their physical and chemical properties and shall place them in their proper position in a general scheme of classification. The paper is based on two seasons of field work in the Oklahoma coal field in 1930 and 1931 and on several weeks spent in visiting most of the operating mines in Arkansas in the same years.

Most of the productive districts of the Arkansas-Oklahoma coal field lie in an east-west trending belt immediately north of the Ouachita Mountains, but the northwestern districts in Oklahoma trend northward toward Kansas. With the exception of the Muskogee district, where very little coal has been mined, the coal beds of the last-mentioned districts are all younger than those of the districts in the east-west trending belt.

Description of the Coal by Districts

Lehigh District, Oklahoma (5).[1]—In the Lehigh district mining has been extensive, but only three or four mines are operating at the present time. Two coal beds of workable thickness, Atoka and Lehigh, are known in this field. The greater part of the mining has been in the Lehigh, or younger bed, and all analyses available for the field have been of that coal. The Lehigh coal is very tough and blocky, the moisture and ash contents are fairly high, 6.5 and 10.7 per cent respectively, the volatile content is 39.5 per cent, fixed carbon 43.4 per cent, B.t.u. value 11,525, and the fuel ratio 1.1 (average 15 analyses). So far as the writer has been able to ascertain, laboratory attempts to coke this coal have been unsuccessful and no attempt has been made to produce coke commercially.

Tulsa District, Oklahoma (1).—All commercial mining in the Tulsa district has been by stripping from the Dawson bed, and production has never been very great. The coal is blocky, contains 6.5 per cent moisture, 36.8 per cent volatile matter, 48.4 per cent fixed carbon, and 8.3 per cent

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† Assistant Geologist, U. S. Geological Survey.

[1] The numbers shown in parentheses after the names of the districts correspond to the numbers of the districts in Fig. 1.

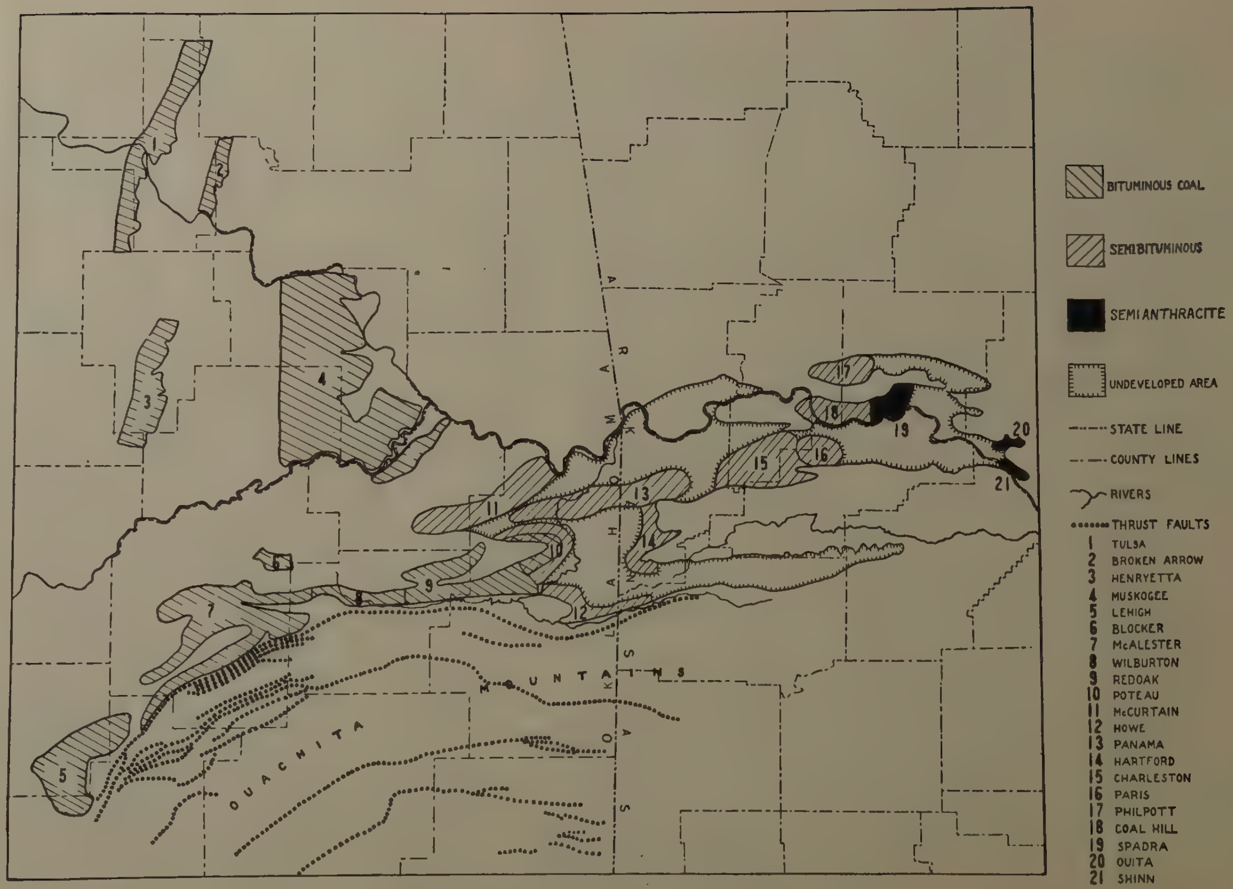

Fig. 1.—Arkansas-Oklahoma coal field.

ash, and has a B.t.u. value of 12,580 and a fuel ratio of 1.3 (average 3 analyses). No attempt has been made to coke this coal, so far as the writer knows.

Blocker District, Oklahoma (6).—In the Blocker district a few small slope mines have produced coal from the Upper Witteville bed. The coal contains 2.4 per cent moisture, 36.8 per cent volatile matter, 51.5 per cent fixed carbon, and 9.3 per cent ash, and has a B.t.u. value of 13,240 and a fuel ratio of 1.4 (only one analysis available). No attempt has been made to coke this coal.

McAlester District, Oklahoma (7).—Coal in the McAlester district has been mined commercially for about 50 years. Five coal beds have been worked, but at the present time only the Lower Hartshorne and McAlester coals are being mined. These beds are of approximately the same quality. A considerable part of the Hartshorne coal mined in this district has been taken from railroad-owned mines and has not gone on the open market, and the bulk of the remainder is marketed under the trade name of "McAlester" coal. Both coals have been used for commercial coking, but since 1905, when coke from other fields began to appear on the western markets, no coking has been attempted. The Lower Hartshorne coal contains 3.7 per cent moisture, 37.8 per cent volatile matter, 51.8 fixed carbon, and 6.6 per cent ash (average 37 analyses). The Upper Hartshorne coal contains 4.5 per cent moisture, 35.4 per cent volatile matter, 53.6 per cent fixed carbon, and 6.5 per cent ash (average 6 analyses). The McAlester coal consists of 3.2 per cent moisture, 36.8 per cent volatile matter, 54.4 per cent fixed carbon, and 5.6 per cent ash (average 31 analyses). The B.t.u. values are: Lower Hartshorne 13,430, Upper Hartshorne 13,225, and McAlester 13,570. The fuel ratios are 1.4, 1.5 and 1.5 respectively.

Wilburton District, Oklahoma (8).—In the Wilburton district three coal beds, Upper and Lower Hartshorne and McAlester, are present. The two Hartshorne coals have been mined in the past, but the McAlester bed is too thin for commercial mining. At present no commercial mining is carried on in this district although the coal is an excellent steam producer and domestic fuel. The chief reason for the abandonment of the field was that the mines were very gaseous and numerous explosions rendered it unprofitable to operate them. The Lower Hartshorne coal contains about 5.6 per cent moisture, 35.6 per cent volatile matter, 53.0 per cent fixed carbon, and 5.8 per cent ash, and has a B.t.u. value of 13,360 and a fuel ratio of 1.5 (average 9 analyses). The Upper Hartshorne coal consists of 3.6 per cent moisture, 37.3 per cent volatile matter, 53.7 per cent fixed carbon and 5.4 per cent ash, and has a B.t.u. value of 13,580 and a fuel ratio of 1.4 (average 9 analyses).

Broken Arrow District, Oklahoma (2).—In the Broken Arrow district some coal is stripped from an unnamed bed near the horizon of the

Henryetta bed. The coal contains 7.6 per cent moisture, 34.5 per cent volatile matter, 50.4 per cent fixed carbon and 7.6 per cent ash. The B.t.u. value is 12,500 and the fuel ratio is 1.5. (Average 3 analyses.)

Henryetta District, Oklahoma (3).—Production in the Henryetta district has increased greatly in the last decade, and in these years the district has been one of the major producers of Oklahoma. The Henryetta coal bed is the only one of commercial importance in this district. The coal is made up of 7.4 per cent moisture, 34.8 per cent volatile matter, 52.7 per cent fixed carbon and 5.1 per cent ash. The B.t.u. value and fuel ratio are 12,960 and 1.5 respectively. (Average 14 analyses.)

Red Oak District, Oklahoma (9).—In the Red Oak district only the Lower Hartshorne coal has been mined. Operations have been fairly extensive in this bed, but the difficulties of mining a steeply dipping and gaseous coal bed have led to the abandonment of all mines. Near Fanshawe, in the eastern part of the district, the coal is somewhat better than the average, but attempts to compete with the low-volatile coals in the extreme eastern part of the state have been unsuccessful. The coal contains 4.7 per cent moisture, 31.6 per cent volatile matter, 59.5 per cent fixed carbon and 4.2 per cent ash, and has a B.t.u. value of 13,910 and a fuel ratio of 1.9 (average 3 analyses).

Muskogee District, Oklahoma (4).—In the Muskogee district the McAlester coal has been mined by stripping at several places, but production has never been very great. The McAlester coal contains 2.3 per cent moisture, 30.5 per cent volatile matter, 60.6 per cent fixed carbon and 6.6 per cent ash. The B.t.u. value is 14,110, and the fuel ratio is 2.0 (only one analysis available).

Poteau District, Oklahoma (10).—In the Poteau district the Cavanal, Upper Witteville and Lower Witteville coal beds have been mined, but at present only the first-named bed is mined. The Lower Witteville coal contains 1.6 per cent moisture, 22.1 per cent volatile matter, 63.0 per cent fixed carbon and 13.3 per cent ash. Its B.t.u. value is 13,180, and its fuel ratio is 2.9. (Average 5 analyses.) This coal is blocky and not noticeably friable in natural exposures.

The Cavanal coal consists of 2.5 per cent moisture, 21.2 per cent volatile matter, 66.3 per cent fixed carbon and 10.0 per cent ash. The coal has a B.t.u. value of 13,700 and a fuel ratio of 3.1 and is somewhat friable. (Average 2 analyses.)

McCurtain District, Oklahoma (11).—The Upper and Lower Hartshorne coal beds have been mined in the McCurtain district, but all available analyses are of the Lower Hartshorne coal. This coal contains 2.9 per cent moisture, 20.7 per cent volatile matter, 69.5 per cent fixed carbon and 6.8 per cent ash. The coal is friable and has a B.t.u. value of 13,990 and a fuel ratio of 3.4. (Average 18 analyses.)

Howe District, Oklahoma-Arkansas (12).—The Lower Hartshorne coal is mined in the Howe district. It consists of 3.3 per cent moisture, 18.8 per cent volatile matter, 70.6 per cent fixed carbon and 7.3 per cent ash. The coal has a B.t.u. value of 13,840 and a fuel ratio of 3.8 and is very friable. (Average 3 analyses.)

Hartford District, Arkansas (14).—Hartshorne coal (upper and lower beds not differentiated) has been mined in the Hartford district. This coal contains 3.3 per cent moisture, 18.2 per cent volatile matter, 69.1 per cent fixed carbon and 9.4 per cent ash. The B.t.u. value is 13,500 and the fuel ratio 3.8, and the coal is extremely friable (average 18 analyses).

Panama District, Oklahoma-Arkansas (13).—In the Panama district mining operations have been carried on in both the Upper and Lower Hartshorne beds, but in available analyses for the Arkansas part of the district no distinction is made between the two. The Lower Hartshorne in the western part of the district contains 2.7 per cent moisture, 16.3 per cent volatile matter, 73.6 per cent fixed carbon and 7.3 per cent ash, and has a B.t.u. value of 14,010 and a fuel ratio of 4.5 (average 7 analyses.) The Upper Hartshorne in the same part of the district is made up of 2.1 per cent moisture, 18.0 per cent volatile matter, 73.0 per cent fixed carbon and 6.9 per cent ash, and has a B.t.u. value of 14,125 and a fuel ratio of 4.1 (average 2 analyses). The Hartshorne (undifferentiated) in the Arkansas part of the district contains 2.4 per cent moisture, 16.7 per cent volatile matter, 69.3 per cent fixed carbon and 11.6 per cent ash; the B.t.u. value is 13,240 and the fuel ratio 4.1 (average 6 analyses). All of these coals are very friable.

Charleston District, Arkansas (15).—In the Charleston district coal is mined from the Charleston bed stratigraphically about 800 ft. above the Hartshorne horizon. Most of the coal has been obtained by stripping. It contains 3.5 per cent moisture, 18.4 per cent volatile matter, 72.7 per cent fixed carbon, 5.4 per cent ash, has a B.t.u. value of 14,350 and a fuel ratio of 4.0 (average 5 analyses). The coal is friable, but because of the method of mining little slack is produced.

Paris District, Arkansas (16).—In the Paris district a coal bed of the same name and lying about 1200 ft. above the Hartshorne horizon is mined extensively. This coal contains 2.4 per cent moisture, 17.2 per cent volatile matter, 70.5 per cent fixed carbon and 9.9 per cent ash; the B.t.u. value is 13,540 and the fuel ratio is 4.1 (average 3 analyses). The coal is friable but since it is mined by an advancing longwall method and the coal falls from its own weight little slack is produced.

Philpott District, Arkansas (17).—Very little coal has been mined in the Philpott district because no thick beds occur there. Some mining has been done in the Hartshorne and a younger bed near the horizon of the Charleston coal. The Philpott district coal contains 2.0 per cent moisture, 16.6 per cent volatile matter, 78.3 per cent fixed carbon, and 3.1 per cent ash. The B.t.u. value and the fuel ratio are 14,870 and 4.7 respectively, and the coal is very friable. (Average 3 analyses.)

Coal Hill District, Arkansas (18).—In the Coal Hill district, the Hartshorne coal has been mined extensively. Here it is only slightly friable and contains 3.4 per cent moisture, 14.5 per cent volatile matter, 73.7 per cent fixed carbon and 8.3 per cent ash. The B.t.u. value is 13,600 and the fuel ratio 5.1. (Average 6 analyses.)

Spadra District, Arkansas (19).—The Spadra district is separated from the Coal Hill district by a normal fault. The Spadra coal, which is at the Hartshorne horizon, is hard coal and has low friability. The coal contains 2.7 per cent moisture, 12.7 per cent volatile matter, 75.6 per cent fixed carbon, and 9.0 per cent ash. The B.t.u. value is 13,660 and the fuel ratio 6.0. (Average 14 analyses.)

Shinn District, Arkansas (21).—In the Shinn district a coal bed at the Hartshorne horizon has been mined extensively. At present only one mine, the Bernice anthracite mine, is operating. The coal is hard and yields but little slack in mining. It contains 2.8 per cent moisture, 11.9 per cent volatile matter, 75.2 per cent fixed carbon and 10.1 per cent ash. The B.t.u. value and fuel ratio are 13,356 and 6.3 respectively. (Only one analysis available.)

Ouita District, Arkansas (20).—In the Ouita district little mining has been done, because of the thinness of the bed and the friability of the coal found there. This coal is found at the Hartshorne horizon but is not definitely known to be the same bed as that mined in the Shinn district about five miles to the south. The coal is rather friable, contains 3.1 per cent moisture, 11.8 per cent volatile matter, 77.6 per cent fixed carbon and 7.5 per cent ash. The B.t.u. value is 13,780, and the fuel ratio is 6.6. (Average 6 analyses.)

Classification of the Coals

Coal from the following districts is classified by the writer as bituminous: Lehigh (5), Tulsa (1), Blocker (6), McAlester (7), Wilburton (8), Broken Arrow (2), Henryetta (3), Red Oak (9), Muskogee (4) and Upper Witteville of the Poteau district (10). This agrees with present marketing practices, but the Lower Witteville coal of the Poteau district is reported to have been marketed as "low-volatile" or "smokeless" coal when it was mined some years ago. However, the abandonment of all mines in this coal bed and the continuation of work at mines in thinner and more steeply dipping coal beds in the Howe district and the Cavanal bed of the Poteau district indicate that the coal from the Upper Witteville bed could not compete in the market with the coals classified as "semi-bituminous" in this paper. The coals classed as bituminous by the writer are blocky and not friable and range in fuel ratio from 1.1 to 2.9, in moisture content from 2.4 to 7.6 and in heating value from 11,525 to

14,110 B.t.u. It should be clearly understood that the minimum and maximum figures given here are not considered by the writer as boundaries for bituminous coal as a whole, but that it is the writer's opinion that the boundaries set for bituminous coal in general and the Arkansas-Oklahoma coal field in particular should be such as to include all figures given above.

The writer classifies coal from the following districts as semibituminous: Cavanal bed of the Poteau district, McCurtain, Howe, Hartford, Panama, Charleston, Paris, Philpott, and Coal Hill. Collier[2] considered the Howe, Hartford, Panama, Charleston and Paris coals bituminous rather than semibituminous. His classification was based on fuel ratios with bituminous coal ranging from 1 to 4½, semibituminous from 4½ to 6½, and semianthracite from 6½ to 8½. In the light of additional data from other fields it is evident that all of these boundaries are placed too high, and that as classified by Collier the coals in the districts mentioned above were placed below their proper rank. In local marketing practice the operators have discarded the name "semibituminous" as unsatisfactory and have substituted the name "semianthracite" for coal of that rank. The writer considers the name "semibituminous" an unsatisfactory term, but is opposed to replacing it by the name "semianthracite," which has a widely accepted use for coal of higher rank than semibituminous. These coals are all friable, contain 3.5 per cent or less moisture, and range in fuel ratio from 3.1 to 5.1, in volatile content from 14.5 to 21.2, and in B.t.u. value from 13,240 to 14,870. It is the writer's opinion that the boundaries for semibituminous coal should be so placed as to include coal with these characteristics.

The writer classifies the coal from the Spadra, Shinn, and Ouita districts as semianthracite, which is in agreement with Collier's conclusions. In local trade practice these coals are called "anthracite," since there the name "semianthracite" has been substituted for semibituminous and all coal above semibituminous rank is called "anthracite." The coals classified herein as semianthracite are hard coals, although one, the Ouita district coal, is somewhat friable. They all range from 6.0 to 6.6 in fuel ratio, from 11.8 to 12.7 in volatile content, and from 75.2 to 77.6 in fixed carbon, and contain 3.1 per cent or less moisture. It is the writer's opinion that the boundaries of semianthracite coal should be so placed as to include coals with the characteristics described above.

DISCUSSION

(*H. J. Rose presiding*)

W. T. Thom, Jr., Princeton, N. J.—The Oklahoma-Arkansas area affords an excellent opportunity for determining whether there is a uniform gradation from bituminous coal on the southwest to anthracite coals at the eastern end of the field. With the apparently gradual eastward increase in the intensity of metamorphic effects suffered by single beds of coal, it should apparently be possible to determine whether natural groups of coals exist, or whether the separation of coal classes will have to be made by arbitrary decision.

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[2] A. J. Collier: The Arkansas Coal Field. U. S. Geol. Survey *Bull.* 326 (1907) 76–86.

T. A. HENDRICKS.—So far as available analyses show the gradation is progressive rather than sharp, but it is steep, or rapid, at two places as between bituminous and semibituminous coal, and the change from semibituminous to anthracite coal is rather sharp. The latter two places referred to are only about one mile apart, but the coals exhibit a difference of 5 or 6 in fuel ratio. There is no apparent structural or other reason for this.

Moisture Determination for Coal Classification

By Edgar Stansfield* and K. C. Gilbart,* Edmonton, Alberta

(New York Meeting, February, 1932)

One of the most striking features of the coal series passing from peat through brown coal, lignite, etc., up to anthracite is the gradual reduction of moisture content with the increased coalification of the original plant material. Thus drained peat from a bog may contain 90 per cent water, brown coal contain 50 per cent, lignite 30 per cent, while a bituminous coal may contain only 1 per cent moisture. Moisture, therefore, is one of the important factors to be considered in coal classification, especially with the lower rank coals. With bituminous and higher rank coals the moisture content is less satisfactory for classification; it may even pass a minimum value and then increase with increasing rank to anthracite and superanthracite.

Coal in an undisturbed seam contains moisture which is an integral part of the coal, but it may also be wet with free, underground water percolating through the seam. Only the former must be considered in evaluating the moisture content of the coal for classification purposes; this will be called the "true moisture" of the coal. When the seam is mined and sampled the sample taken may show the true moisture; on the contrary, it may contain in addition free mine water or, especially in dry, well ventilated mines, it may contain less than the true moisture.

Coal contains unstable organic compounds which may decompose, with the production of moisture, when the coal is exposed to air at ordinary temperatures, or is heated with or without exposure to air. The potential moisture of a coal therefore is higher than its actual moisture. The lower the rank of the coal, the greater its instability, as a rule, and the greater the care necessary to avoid oxidation or decomposition in any investigation of moisture content.

The majority of coal workers have preferred, on account of the difficulty in evaluating the true moisture of a coal, and for other reasons, to classify coals by analyses calculated to the dry-coal basis. A number of workers familiar with lower rank coals have recently insisted that the classification of such coals can be satisfactory only if analyses on the moist-coal basis are employed. The moisture-holding capacity of coal is therefore intimately connected with coal classification, and in this paper a procedure is outlined for determining the true moisture of a coal.

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Attempts have been made to evaluate the moisture-holding capacity of a coal by determining the moisture retained after a partial drying under standardized conditions. This process, commonly known as "air-drying," will also be discussed.

Partial Drying or Air-drying of Coal

In the laboratory handling of coal it is necessary to subject the sample to a preliminary, partial drying, commonly called "air-drying," to bring the coal to such a state of equilibrium with the humidity of the air in the laboratory that no appreciable change will take place during manipulation. This treatment is also advantageous because the original moist coal would often give trouble if subjected to fine grinding.

The simplest method of air-drying is to expose the crushed coal, in shallow trays, to the air of the laboratory. The A. S. T. M. method for air-drying[1] specifies that the crushed coal shall be exposed in a shallow tray, in a current of air heated 10° to 15° C. above room temperature, until the loss in weight is less than 0.1 per cent per hour. The U. S. Bureau of Mines air-dries in air at 30° C.

A prolonged attempt has been made in Canada to make the test serve the dual purpose of preparing the sample for laboratory handling and of evaluating the moisture-holding capacity of the coal. For this reason the coal has been air-dried in an atmosphere regulated to a standard relative humidity. The first method,[2] developed in 1907, consisted of exposing crushed coal (through ¼ in.) in shallow trays in a box which also held trays containing a solution of calcium chloride of 1.30 sp. gr. This solution, at ordinary temperatures, has a vapor pressure about 60 per cent of that of water at the same temperature. The coal was weighed from time to time until a minimum weight was recorded. This was a prolonged procedure, taking up to six weeks in extreme cases, and the method was abandoned in 1910. The 60 per cent relative humidity given by this calcium chloride solution, however, was employed in subsequent standard humidity methods.

A second method was developed in 1923 when an air-drying apparatus was constructed in these laboratories.[3] In this method the crushed coal was dried in a rapid stream of air of 60 per cent humidity. The humidity was controlled by causing the circulating air to pass up a tube in which the calcium chloride solution ran down a number of lamp wicks. In this method 48 hr. was sufficient for the attainment of practical equilibrium. The apparatus employed was repeatedly improved up to 1930. The final model is shown in Appendix I.

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[1] Laboratory sampling and analysis of coal and coke. Amer. Soc. Test. Mat. D 271–29.

[2] Investigations of the Coals of Canada. Dept. of Mines, Ottawa, *Rept.* 83 (1912) 130.

[3] E. Stansfield: A Standardized Method for Air-Drying Coal. *Trans.* Can. Inst. Min. and Met. (1923) **26,** 292; also, Research Council of Alberta, Fourth Annual Report (1923) 39.

This apparatus was unsatisfactory in that there was no control of temperature and, as air was used, the coal was liable to oxidation. Control of temperature is essential if the accuracy required for classification purposes is to be attained, since it has been found that the change with temperature of the vapor pressure of water in coal is not the same, even through the small range of laboratory temperatures, as the change with temperature of the vapor pressure of the calcium chloride solution. Therefore a new apparatus was constructed, early in 1931, in which a uniform temperature of 30° C. is maintained and methane (natural gas) of 60 per cent humidity is circulated. Equilibrium is reached, as before, within 48 hr. This apparatus is also described in Appendix I.

A vacuum desiccator procedure, for drying at constant temperature to various definite humidities, was also adopted in 1931. This is described in a later section.

The moisture retained after air-drying at 60 per cent humidity, by the method developed in 1923, has been employed for coal classification by the Canadian Department of Customs and Excise.[4] The relation between true moisture and air-dry moisture will be discussed later, but it should be stated that the latter values have not proved satisfactory for scientific classification of coal and are, as far as the writers are aware, no longer recommended for classification purposes by any coal workers.

DETERMINATION OF RESIDUAL AND TOTAL MOISTURE IN COAL

Any method used for determining the total moisture in coal could also be used for determining the residual moisture in air-dry coal, but some of these are seldom employed except for determining the total moisture in the sample prior to grinding. Several methods are outlined below for reference in the subsequent discussion on the possible errors of moisture determinations.

A. *Standard A. S. T. M. Method.*—One gram of the ground coal is dried for 1 hr., in an oven at a temperature of 104° to 110° C., in a brisk current of dried air. The loss in weight is assumed to equal the moisture driven off.

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[4] *Appraisers Bull.* 2814, Ottawa (Aug. 25, 1923). "Lignite, or lignitic coals are to be defined as those grades of coal having on the air-dried basis not less than 6 per cent moisture content.

"Both the brown lignites and the black lignites also known as 'subbituminous' come within the class of lignitic coals, and, therefore, may all be termed 'lignites.'

"The standard method for air-drying referred to above is that recently adopted by the Provincial Government of Alberta and by the Department of Mines, Ottawa, and consists of exposing the crushed coal in shallow layers at room temperature in an atmosphere of approximately 60 per cent humidity."

B. *Modified Oven Method.*—A modification of the above, approved[5] by the B. E. S. A. for oxidizable coals but not approved by the A. S. T. M., is the use of a current of dried neutral gas such as nitrogen (B. E. S. A.), carbon dioxide, or methane (natural gas), instead of dried air. The coal is cooled in an evacuated desiccator to remove occluded gas.

C. *Vacuum Oven Method.*—B. E. S. A. suggests,[6] as a second alternative for coals liable to oxidation, the maintenance of a vacuum suction of 51 to 64 cm. of mercury in the oven.

D. *Vacuum Oven Method with Gas Circulation.*—This is a method employed in these laboratories for determining the moisture in unground samples. A 5-gram portion is heated at 105° to 110° C. for 3 hr. in a Freas vacuum oven, through which a slow current of dried methane (natural gas) is passed but in which a vacuum of about 56 cm. is maintained by a large pump. The circulation of gas is believed to reduce risk of oxidation from air leaks and to assist in the quick removal of liberated moisture. The coal is cooled in an evacuated desiccator.

E. *Distillation Method.*—A method frequently employed for determining total moisture in crushed coal consists in adding to the coal an organic liquid nonmiscible with water, distilling the moisture off with the liquid, and collecting and measuring this water in a graduated vessel. Xylene and toluene are the liquids commonly employed, and this method (with toluene) is approved by B. E. S. A. for coals liable to oxidation. Many forms of apparatus have been employed for this method; the one employed in these laboratories is shown in Appendix II. A 50-gram sample is usually treated.

F. *Dehydration Over Desiccants.*—This method is commonly used only for determination of total moisture. The sample is left in a desiccator at room temperatures over concentrated sulfuric acid, "dehydrite," or other desiccant. The removal of moisture from the coal can be notably expedited, and oxidation can be prevented, by employing an evacuated desiccator.

G. *Calculation of Total Moisture.*—The routine determination of total moisture is made by calculation from the loss on air-drying and the determined residual moisture.

The more important possible errors in the moisture determination are oxidation of the coal, decomposition of the coal, and failure to completely remove the moisture.

Oxidation

Most coals absorb oxygen at a notable rate at room temperatures and rapidly at 110° C., and thereby gain in weight. The oxidation of the coal, however, causes the formation of additional water. When moisture determinations are made by method A, the gain in weight by oxidation reduces the measured loss in weight so that the determined value is too small. This error, which is reduced to a low value in the other methods, is serious with low-rank coals but notable even with some bituminous coals. Tests were made in Ottawa[7] in 1911 on six coals, five British and one South American, sent out in connection with an investigation on moisture for the Eighth International Congress of Applied Chemistry. These showed, for example, that if the coals were dried for 2 hr. in a current of neutral gas and then reheated for 1 hr. in air, there was invariably a gain in weight during the reheating. This ranged from 0.03 to 0.19 per cent with an average value of 0.1 per cent. The moisture in the coals ranged from 3 to 10 per cent. A dry, low-rank coal has been known to ignite when heated in a current of air in the drying oven.

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[5] British Standard Specification for the Sampling and Analysis of Coal 420, British Eng. Stds. Assn. (1931).

[6] British Standard Specification for the Sampling and Analysis of Coal 420, British Eng. Stds. Assn. (1931).

The oxidation of coal during the moisture determination, as above, is accompanied not only by gain in weight but, presumably, also by formation of fresh water. In laboratory practice the former predominates and therefore determinations made in air (method A) give too low a result. Oxidation of the sample, prior to the determination, results in gain in weight as well as in the formation of additional water. When ground samples are stored in the laboratory in tightly stoppered glass bottles, and two moisture determinations are made a week or two apart, the latter value is almost invariably the higher even though the coal would lose moisture if the bottle were unstoppered.

A series of 19 ground samples of air-dried coal, tested earlier in the regular routine work of the laboratory, were retested for moisture content. The samples had been stored in bottles with ground-glass stoppers in the balance room for periods ranging from 12 to 36 days, and the bottles had not been disturbed since the earlier determination. It was found that the moisture content had risen in every case, the increase varying from 0.08 per cent on a bituminous coal with 0.5 per cent water, stored for 33 days, to 1.32 per cent on a coal with 7 per cent of water, stored for 34 days. A sample with 16 per cent moisture gained 0.20 per cent in 12 days. The moisture increase tended to increase with period of storing and to be higher in the lower rank coals, but complete regularity could not be expected on account of differences in the fit of the stoppers. Another set of 18 similar coal samples was retested. In this case the bottles had been opened in the meantime to withdraw portions for ultimate analysis. The average increase was less, as was to be expected, 0.23 per cent in this series compared with 0.40 per cent in the former series. The changes ranged from a 0.22 per cent loss with a bituminous coal stored 27 days to a 0.66 per cent gain with a 15 per cent moisture coal stored 24 days. A 20 per cent moisture coal gained 0.17 per cent in 13 days. The 74 moisture determinations were made in duplicate, and the average difference between duplicates was 0.08 per cent.

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[7] Summary Report, Mines Branch, Dept. of Mines, Canada (1911) 123.

This formation of water at ordinary temperatures is negligible when the coal is protected from oxidation. This was shown by the following experiment:[8] Two samples of the same moist coal, between 8 and 14 mesh, were exposed for about one month to a slow stream of dry air for one sample and of dry natural gas for the other. The change in weight of the samples, water carried off in the gas stream and residual moisture were determined in each case. The results follow:

	In Air	**In Methane**
Loss in weight of coal + residual moisture, per cent......	25.89	26.09
Collected moisture + residual moisture, per cent........	29.25	26.49
Moisture determined in original coal, per cent...........	26.35	26.35
Water of oxidation, per cent..........................	2.90	0.14

In the writers' opinion, errors due to oxidation can be avoided only by the complete analysis of the sample within a short period after grinding, and by the exclusion of air in the moisture determination. The use of evacuated containers, or atmospheres of neutral gas, is recommended for the storage and grinding of samples of coal, particularly of low-rank coal.

Decomposition

Many workers have studied possible errors in moisture determinations due to decomposition of the coal. The committee on investigation of moisture for the Eighth International Congress of Applied Chemistry reported[9] evidence that no appreciable amounts of hydrocarbons are lost by coal even when heated for longer times and at higher temperatures than in the moisture determination. This point was checked also in these laboratories in 1925, at 106° C., in a current of nitrogen. The nitrogen leaving the coal passed through a combustion apparatus, as used for ultimate analyses, and the resulting carbon dioxide was weighed. Crow's Nest coal, moisture 1 per cent, gave 0.07 per cent carbon. Drumheller coal, moisture 18 per cent, gave (1) 0.06, (2) 0.14, (3) 0.11 per cent carbon. Edmonton coal, moisture 24 per cent, gave (1) 0.12, (2) 0.15 per cent carbon. If the carbon is assumed to be given off as methane the maximum loss, with twice the normal time of heating, would be 0.2

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[8] W. P. Campbell: M. Sc. thesis, University of Alberta, 1925.

[9] *Original Communications*, 8th Intl. Cong. of Applied Chem., **25**, 82.

per cent. Actually the values found were probably less than the possible errors of the experiment.

Incomplete Loss of Moisture

It has been claimed that the moisture is not completely driven off under the conditions of the determination. This is a very difficult point to prove, since any more strenuous treatment, or even temporary exposure to air, may result in the production of fresh water. In method B a rapid stream of neutral gas is required to give consistent results, but under these conditions no notable extra moisture is given off during a second hour of heating.

No systematic comparison has yet been made in these laboratories of the different methods outlined above, but some figures are available. Nine samples of coal, crushed through 4 mesh and ranging in moisture content from 10 to 29 per cent, were tested in routine work as follows: One bottleful of each sample was partly dried in methane in the air-drying apparatus, ground, the residual moisture determined by method B, and the total moisture calculated; a second bottleful was crushed through 14 mesh, and some 20 portions of 5 g. each were separately air-dried in vacuo to different humidities, the residual moisture in each portion being determined by method D, and the total moisture calculated and averaged. The differences in moistures found by the two procedures never exceeded 0.5 per cent and the algebraic average difference was only 0.05 per cent. Other portions of the second bottles referred to above were tested by method E, using xylene (b.pt. 139° C). This method gave results which averaged 0.29 per cent higher than vacuum air-drying followed by method D. Comparisons between the use of xylene and toluene (b.pt. 110.7° C.) for method E gave differences, in four samples tested, ranging from 0.03 per cent with a 1 per cent moisture coal to 0.70 per cent with a 29 per cent moisture coal; the value with xylene being always the higher. The boiling point of xylene appears to be too high to give satisfactory checks with methods B and D on high-moisture coals; the use of toluene would seem preferable.

The reasonably good agreement found between the different methods, when air is excluded and the temperature does not exceed 110° C., suggests that the values found by careful determinations do give the total actual moisture content of the sample as tested. In general it is submitted that "moisture" in coal, like other proximate analyses of coal, is admittedly a matter of definition and specification. Nevertheless, if reasonable precautions are taken to avoid oxidation, it is claimed that the determination is as accurate as most other coal determinations and is capable of close duplication in different laboratories. Moreover, it does evaluate, with essential accuracy, the actual moisture in the sample tested.

DETERMINATION OF TRUE MOISTURE AND MOISTURE-HOLDING CAPACITY OF COAL

Several workers in recent years have studied the relation between the moisture in coal, charcoal and other solids and the vapor pressure of such moisture. Various methods have been employed for such work on coal, notably with the use of desiccators and constant-humidity solutions.[10] Some of the workers have employed evacuated desiccators to expedite attainment of equilibrium and to prevent oxidation. Such work has been carried out mainly to study the mechanism by which water is held, and the hysteresis shown in the repeated dehydration and hydration of the material.

The writers, in the course of a prolonged investigation of the air-drying of coals for characterization and possible classification purposes, have developed a laboratory procedure for the rapid, routine determination of the moisture-holding capacity of a coal and the evaluation of the true moisture of the coal as it occurs in the seam.

In the earliest air-drying apparatus referred to before, where stagnant air was employed, weeks were required for the attainment of equilibrium. In the later apparatus, with a rapid circulation of air or gas, two days were found to be amply sufficient. At the beginning of the recent investigations coal was placed on the pan of a recording balance in a stream of methane of regulated humidity and temperature. Saturated salt solutions were used for regulation of humidity. A coal with 24 per cent moisture attained equilibrium, with loss of 4 per cent moisture, in 12 hr. at 75 per cent humidity and 30° C.

This method was found to be cumbersome and permitted the use of only one humidity at a time. The vacuum desiccator method was then found to be at least as rapid in the attainment of equilibrium, far simpler, and to permit the simultaneous test at a number of different humidities. The equipment consists of a constant-temperature chamber provided with heaters and a fan, maintained at 30° C. ± 0.05°; some nine or ten vacuum desiccators with well ground stopcocks and aluminum capsules 2 in. dia. and ⅞ in. deep, with well fitting lids. The desired humidity is maintained in the desiccators by placing "dehydrite" in one and saturated solutions, with an excess of the salt, in each of the others. The following solutions have so far been employed: lithium chloride (12 per cent relative humidity at 30° C.); potassium acetate (20 per cent); potassium carbonate (44 per cent); ammonium nitrate (60 per cent); sodium chloride (75 per cent); barium chloride (88 per cent); sodium tartrate (91 per cent) and potassium sulfate (96 per cent). The humidities of some of these solutions are not stated at 30° C. in the International Critical Tables, and the values of the actual solutions used are shortly to be determined, therefore the approximate values stated in this paper are subject to correction. The desiccators are evacuated by means of a Hy-vac pump and the pressures throughout the experiment are indicated by means of a small vacuum gage in each desiccator; these pressures are almost identical with the vapor pressure of the particular solution. The desiccators are kept evacuated between experiments to avoid absorption of air by the solutions.

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[10] H. C. Porter and O. C. Ralston: Some Properties of the Water in Coal. U. S. Bur. Mines *Tech. Paper* 113 (1916).

I. Lavine and A. W. Gauger: Studies in the Development of Dakota Lignite. *Ind. & Eng. Chem.* (1930) **22,** 1226.

The procedure employed is to crush the fresh coal sample to pass a 14-mesh screen. Portions of about 5 g. are placed in each of some 20 capsules, which are at once covered and accurately weighed. Two capsules, with covers removed, are then placed in each of the desiccators. The desiccators are evacuated and placed in the constant-temperature chamber. After 48 hr. the capsules are removed, covered and reweighed. The residual moisture is then determined by method D. Although the duplicates are treated together in the desiccators they are run in separate batches in the determination of residual moisture.

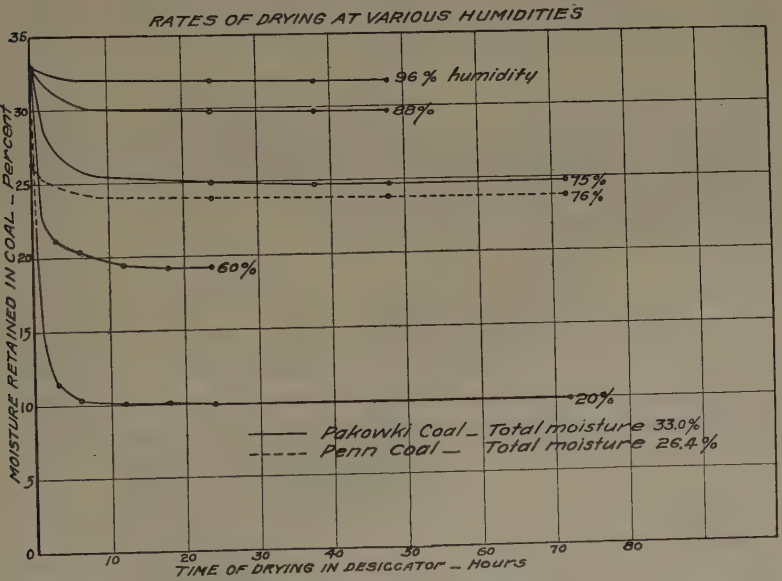

FIG. 1.—RATES OF DRYING AT VARIOUS HUMIDITIES.

The time of 48 hr. in the desiccators was chosen after a number of preliminary tests of the rate of drying represented in Fig. 1, and of comparisons of 24-hr. and 48-hr. drying shown in Table 1. The values for retained moisture show that equilibrium was attained in 24 hours.

TABLE 1.—*Comparison of One-day and Two-day Drying*[a]

Solution Humidity, Per Cent	One-day Test		Two-day Test	
	Residual Moisture, Per Cent	Total Moisture, Per Cent	Residual Moisture, Per Cent	Total Moisture, Per Cent
20	4.91	11.11	4.94	11.19
	4.90	11.10	4.93	11.15
60	8.39	11.12	8.41	11.22
	8.38	11.10	8.35	11.21
75	9.10	11.03	9.11	11.21
	9.18	11.05	9.07	11.21
96	10.00	11.00	10.02	11.09
	10.00	11.00	10.03	11.13
Average.....	8.11		8.11	

[a] This sample of Lethbridge coal had been stored under water and contained surface moisture; 22 samples for the 48-hr. test were weighed out on Sept. 14, and 8 for the 24-hr. test on Sept. 16. The total moisture dropped from 11.2 when the first sample was weighed to 11.0 for the last sample.

One advantage of the evacuation of the desiccator is shown by comparison with the periods of 40 days and more employed by Lavine and Gauger.[11] It also accomplishes the necessary avoidance of oxidation.

The differences in retained moisture found in duplicate determinations are shown in Table 2, which gives the results obtained with the highest moisture coal tested. The total moisture decreased from 35.0 to 34.9 per cent with successive openings of the sample bottle.

The method may be criticized because only 5-gram portions are taken of an unground sample. The close checking shown in the table, and the smoothness of the following curves, indicate that the resulting inaccuracies are not serious.

The retained moisture of the coal plotted against the relative humidity of the desiccator solution usually gives regular curves of the type shown in Fig. 2. Three high-moisture coals, among the 21 coals so far tested, have given irregular curves as shown in Fig. 3. Two of these coals had been stored under water for nearly two years, but were centrifuged before testing. Repeat tests on samples taken at the same time, but stored dry in quart sealers, gave similar curves, but with a tendency to slightly higher moisture values suggestive of oxidation during storage.

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[11] Lavine and Gauger: Studies in the Development of Dakota Lignite. *Ind. & Eng. Chem.* (1930) **22**, 1228.

TABLE 2.—*Results Obtained with Pakowki Coal*

Solution Humidity, Per Cent	Loss in Desiccator, Per Cent	Retained Moisture, Per Cent	Difference between Duplicates	Total Moisture, Per Cent
96	3.59 3.58	32.57 32.58	0.01	34.98 35.00
91	7.55 7.30	29.54 29.75	0.21	34.88 34.88
88	7.04 7.02	30.03 30.05	0.02	34.96 34.96
75	14.30 14.17	24.12 24.14	0.02	34.96 34.90
60	19.63 19.56	19.10 19.12	0.02	34.98 34.96
44	23.69 23.71	14.66 14.66	0.00	34.90 34.90
20	27.44 27.50	10.25 10.20	0.05	34.90 34.91
12	29.27 29.26	8.03 8.02	0.01	34.96 34.95
Average.....			0.04	34.94

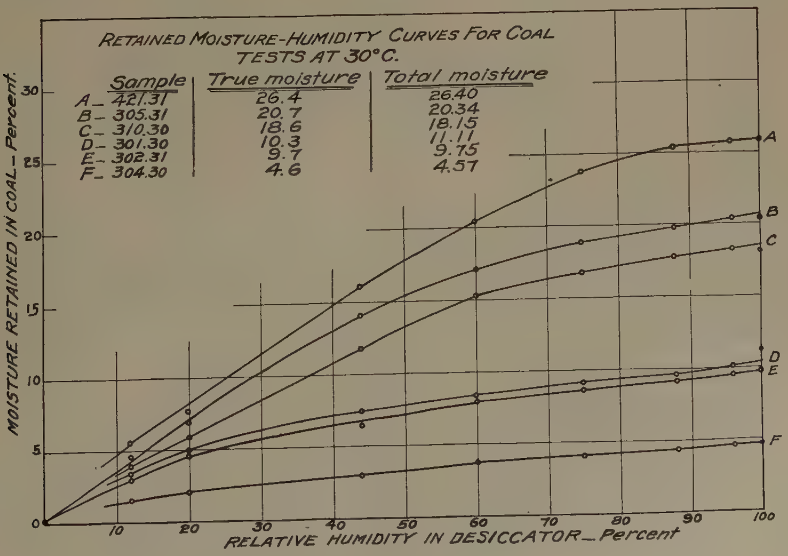

FIG. 2.—RETAINED MOISTURE-HUMIDITY CURVES FOR COAL.

TRUE MOISTURE

The curves prepared as described show moistures retained under conditions up to 96 per cent humidity. It is submitted that if such a curve for a freshly mined coal is extrapolated to cut the 100 per cent humidity line the point of intersection gives the true moisture for the coal in seam, and that this is the moisture value which should be used for classification purposes. It is assumed here that the true moisture of the coal is the minimum amount of water in the fresh coal which can exert the same vapor pressure as free water at the same temperature. Any additional water has the same vapor pressure as free water, but any removal of water below this point is at once indicated by a reduction of vapor pressure. In Fig. 2, for curves *A*, *E* and *F* the point of intersection coincides with the average total moisture of the sample tested. Curve *D* was for coal which appeared to have free surface moisture when tested and in this case the point of intersection (true moisture) was lower than the average total moisture. Curves *B* and *C*, on the contrary, were for coals which had apparently been slightly dried in the mine, or in handling, as in these cases the point of intersection was above the average total moisture.

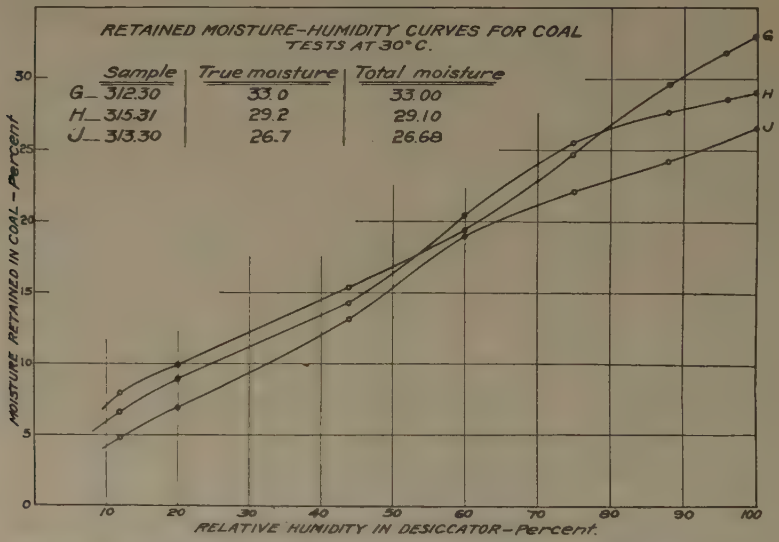

FIG. 3.—IRREGULAR, RETAINED MOISTURE-HUMIDITY CURVES FOR COAL.

This method will not give the true moisture for a partly dried sample if the hysteresis effect, already referred to, is serious. Tests showed that if the sample is dried only in atmospheres of 75 per cent humidity, or over, the coal can reabsorb moisture to practically its original value. The results of these tests, shown in Table 3, are also interesting because

TABLE 3.—*Dehydration and Rehydration of Drumheller Coal*

Regular Test of Coal		Dehydration (48 Hr.)		Rehydration (72 Hr.)		Recovery, Per Cent
Humidity, Per Cent	Retained Moisture, Per Cent	Humidity, Per Cent	Retained Moisture, Per Cent	Humidity, Per Cent	Retained Moisture, Per Cent	
96	20.37	75	18.93	96	20.22	99.3
		60	17.21	96	20.17	99.0
		20	7.44	96	20.13	98.8
		12	5.03	96	20.11	98.7
88	19.75	75	18.93	88	19.95	101.0
		20	7.51		19.15	97.0
75	18.84	60	17.24	75	18.46	98.0
		20	7.49		17.23	91.5
60	17.07	20	7.47	60	14.33	84.0
		12	5.30		14.18	83.1

TABLE 4.—*True Moisture, Total Moisture and Air-dry Moisture at* 60 *Per Cent Humidity*

Sample No.	True Moisture, Per Cent	Average Total Moisture, Per Cent	Air-dry Moisture, Per Cent	Air-dry Moisture as Percentage of True Moisture
434–31	3.8	3.50	3.00	79
304–30	4.6	4.57	3.69	80
301–31	9.6	9.58	7.92	83
302–31	9.7	9.75	7.87	81
301–30	10.3	11.11	8.38	81
16–31	12.2	12.42	10.36	85
9–31	12.9	12.35	10.52	82
308–30	12.9	12.60	11.32	88
14–31	18.5	18.87	14.40	79
310–30	18.6	18.15	15.21	82
310–31	19.0	18.78	16.11	85
306–31	19.3	18.82	16.35	85
308–31	19.8	19.60	16.82	85
305–31	20.7	20.34	17.07	83
13–31	22.1	22.12	15.07	68
421–31	26.4	26.40	20.46	78
313–30	26.7	26.68	19.02	74
313–31	26.9	26.94	20.32	76
314–31	29.0	29.00	20.31	70
315–31	29.2	29.10	20.38	70
312–30	33.0	33.00	19.25	58

the recoveries to high humidities were better than to medium humidities. Seam samples taken with reasonable care should never have lost much moisture, so that the true moisture values of the coal, determined by the method advocated, should be fairly accurate.

Table 4 gives values, for the 21 coals tested, of true moisture taken from the curves for comparison with the average total moisture of the portions tested and with the moisture retained after drying at 60 per cent humidity. The erratic nature of the percentages shown in the last column indicates why the attempts to use the moisture retained after air-drying at 60 per cent humidity were abandoned as unsatisfactory. The regular air-drying procedure gave values for these coals in close agreement with 60 per cent humidity values by the evacuated desiccator method.

In conclusion, the writers are of the opinion that lower rank coals can only be satisfactorily classified by analyses which include the moisture of the coal, and that the true moisture of the coal in the seam is the most satisfactory value for this purpose. The procedure described permits a rapid, routine determination of the true moisture of the coal even though the sample contains extraneous water or has become slightly dried.

Appendix I.—Two Forms of Air-drying Apparatus for Coal

An early form of the air-drying apparatus was described in 1923. Fig. 4 shows the appearance of the apparatus as used in 1930, and Fig. 5 gives a diagrammatic representation. The general principles are the same as those of the earlier model. The coal is dried in trays in the upper box while a fan in the lower box maintains a rapid circulation of air. The humidity of the air is controlled by a flow of calcium chloride solution, of 1.30 sp. gr., down the many lamp wicks in the uptake and downtake air pipes. The solution is lifted to the top of the apparatus, from the large stock bottle in which a hydrometer floats, by means of an air-lift pump. The air pump to operate this is lubricated by an automatic oil circulation. Leaks inward are reduced by the escape of the fresh, moistened and regulated air introduced through the pump.

A new apparatus was constructed in 1930 in which both temperature and humidity are regulated (30° C. and 60 per cent) and a neutral atmosphere is maintained. This is sketched in outside appearance and diagrammatic section in Fig. 6. The trays of coal are interspersed with large trays over which calcium chloride solution flows in a zigzag fashion, falling from tray to tray down into the reservoir in the base. Methane gas (natural gas) is circulated by means of an exterior, Crowell blower. The stream of gas entering the box passes under the solution in a sump in the reservoir, and blows a stream of the liquid up to the top of the box, where it is caught and runs into the top solution tray. This serves the double purpose of lifting the solution and of bringing the gas into intimate contact with the solution and thus regulating the humidity in the chamber. The gas, freed from spray, enters the coal chamber near the bottom, circulates over the coal trays, passes out again near the top, and returns to the suction side of the blower. Additional gas can be introduced as required to remove air. The box is metal lined and air-tight, the removable lid has a vaseline seal. The coal trays can be introduced and removed through small circular doors without serious disturbance of the system. The specific gravity of the solution, and wet and dry-bulb temperatures of the gas, can be ascertained without opening the box. The solution reservoir and air jacket are heated by thermostatically controlled electric light bulbs. The drier takes 24 coal trays, each holding 125 g. of crushed coal. The solution reservoir holds about 10 gal. The metal linings of the box, partitions, trays, etc., are made of galvanized iron. This iron is painted with asphalt paint wherever there is any possibility of its coming in contact with spray or solution. The blower has a free-air capacity of 9 cu. ft. per minute.

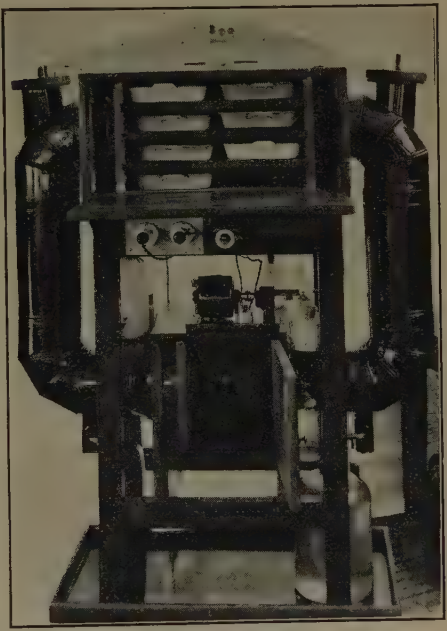

FIG. 4.—CONSTANT HUMIDITY AIR-DRYING APPARATUS.

APPENDIX II.—AN APPARATUS FOR MOISTURE DETERMINATIONS BY DISTILLATION METHOD

BY EDGAR STANSFIELD AND R. G. BREWER*

The xylene distillation method for moisture determinations was adapted for use with coals by Dr. P. Schläpfer. Published references were made to his method in 1912 and details were published[12] in 1914. The coal was distilled with xylene and the distillate received in a vessel with a narrow stem at the base in which the volume of water collected could be measured. Subsequently the method was modified, notably by using a small collection vessel so arranged that the excess xylene ran back into the distilling flask. Well known forms of this apparatus

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* Fuels Division, Research Council of Alberta.

[12] P. Schläpfer: *Ztsch. f. angew. Chem.* (1914) **27,** 52–56.

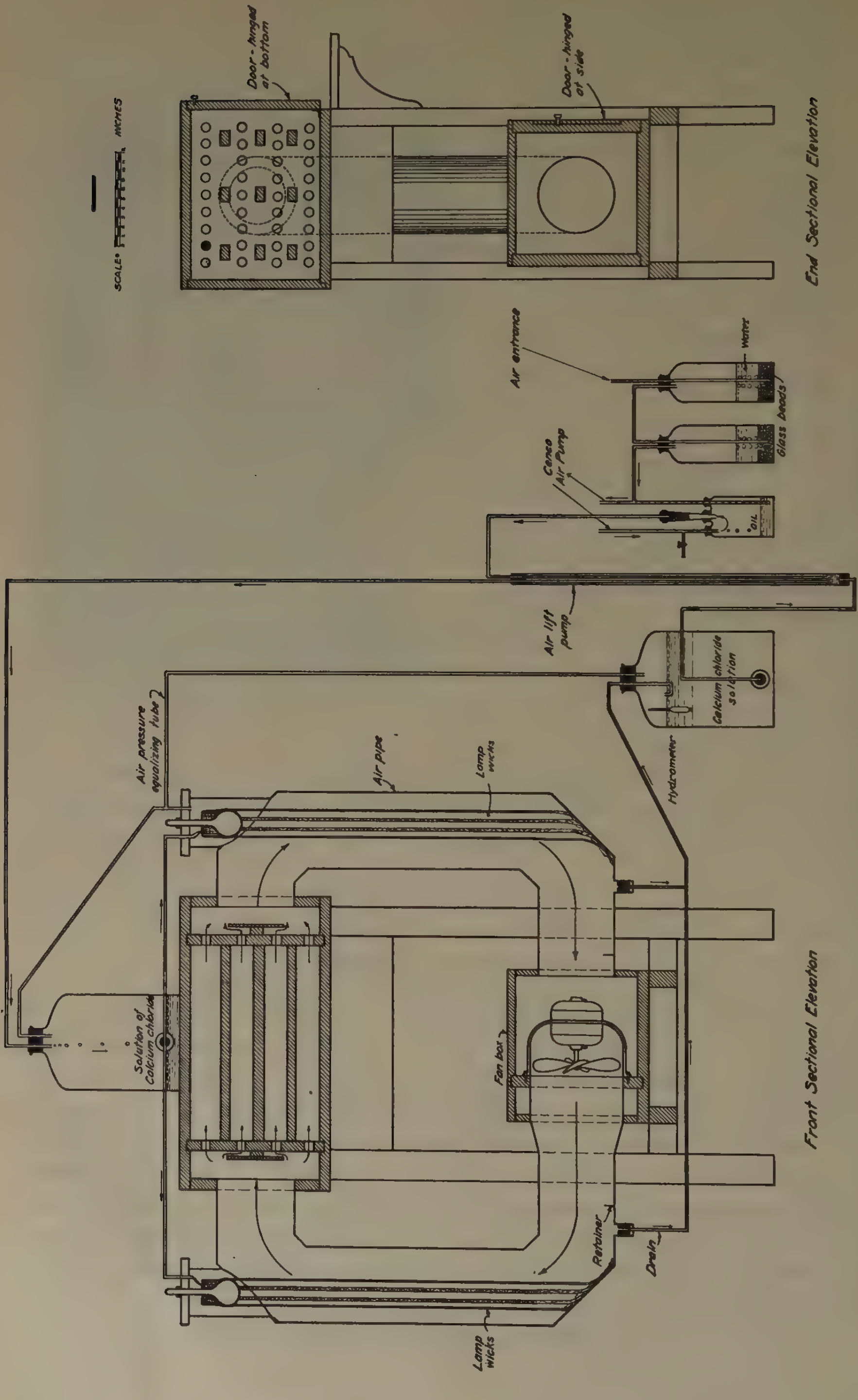

FIG. 5.—CONSTANT HUMIDITY AIR-DRYING APPARATUS.

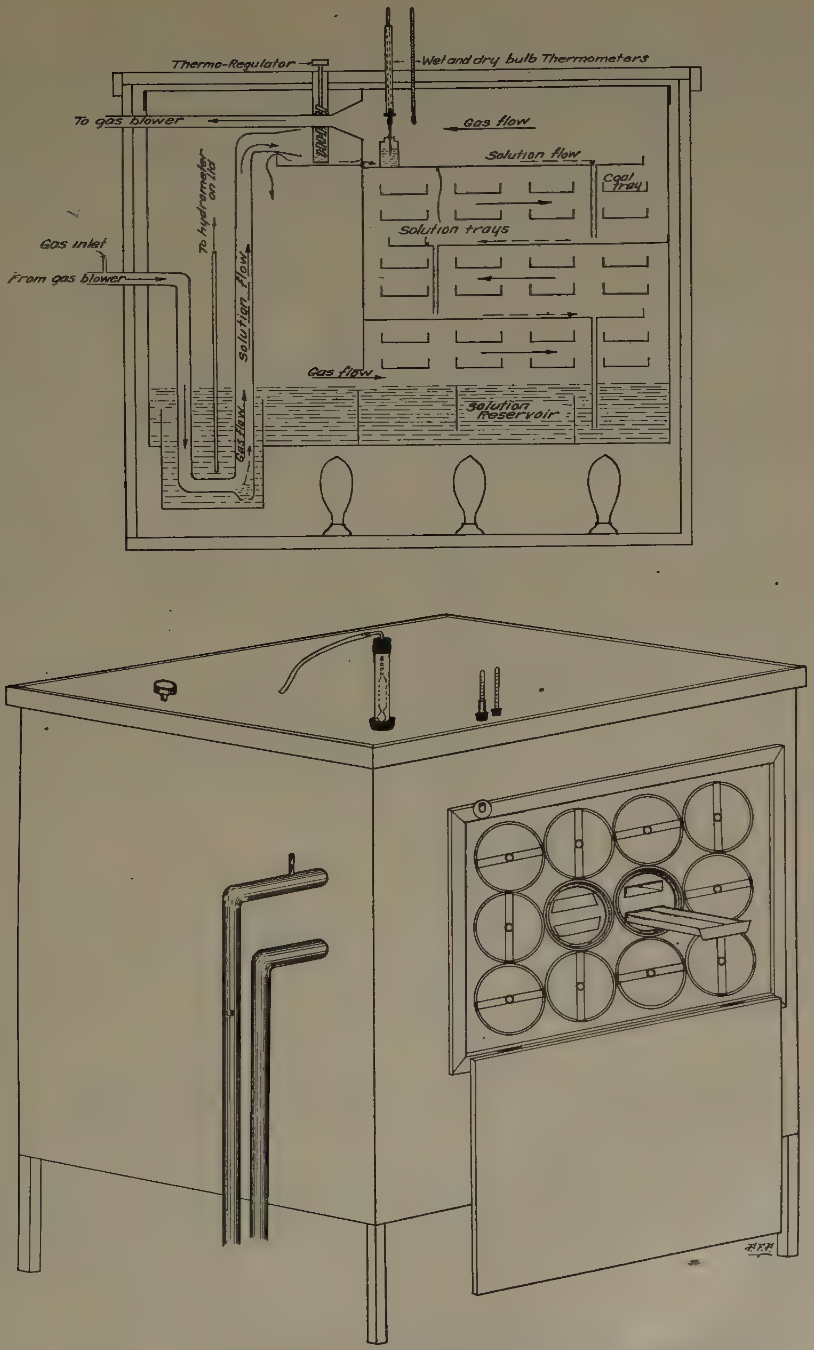

FIG. 6.—CONSTANT TEMPERATURE AND HUMIDITY AIR-DRYING APPARATUS.

are the A. S. T. M.[13] and the B. E. S. A.[14] In these forms the volatile liquid (xylene or toluene is commonly used) and the water vapor pass up into a reflux condenser where they are condensed and run down into a graduated receiver. The water sinks to the bottom and excess xylene overflows back into the distilling flask. The most serious difficulties encountered are the tendency for the condensed water to adhere to the walls of the condenser and to the upper walls of the receiver, and the formation of an emulsion of xylene and water.

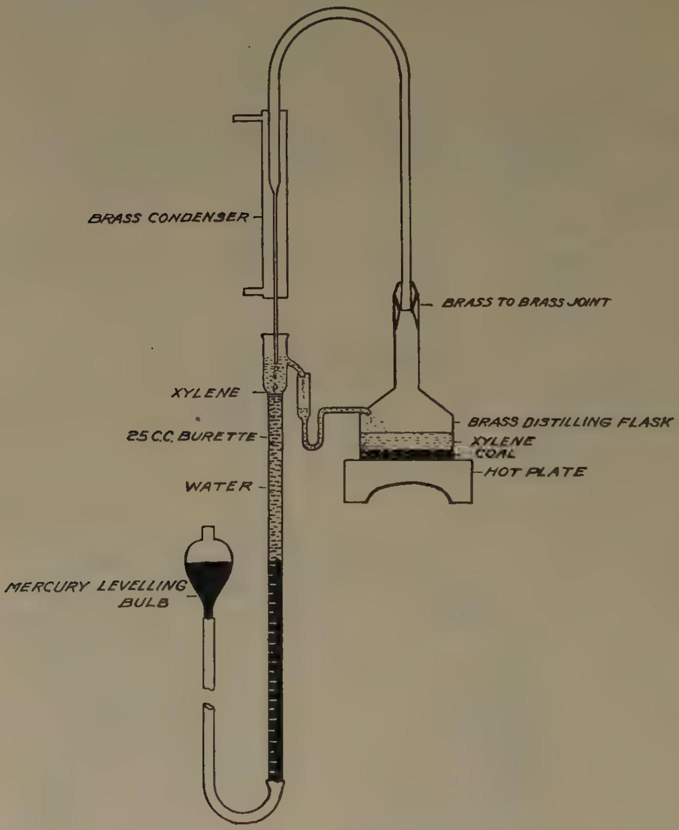

FIG. 7.—DISTILLATION APPARATUS FOR MOISTURE DETERMINATIONS.

The apparatus constructed in this laboratory (Fig. 7) has a single-direction, vertical condenser, with a narrow and tapered condenser tube down which the condenser liquid flows rapidly and thus hinders the adhesion of water to the walls. The retention of moisture on the walls of the receiver is prevented by the enlarged section at the top, and by the use of an adjustable, mercury-levelling bulb which makes it possible to keep the water-xylene surface always near the top of the narrow part of the burette. Moreover, the volume of emulsion, and consequently the loss of water in emulsion or in solution in the xylene, can in this way be kept to a constant minimum. Raising and lowering the mercury assists in the collection of water drops.

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[13] Standard Method for Water in Petroleum Products and Other Bituminous Materials, Amer. Soc. Test. Mat. D 95–30.

[14] Standard Specification for the Sampling and Analysis of Coal 420, British. Eng. Stds. Assn. (1931) 17, 18.

For this test 100 g. of coal can be taken and 150 c.c. of xylene or toluene. The apparatus is connected up and the distillation conducted at a brisk pace. The condenser tube is given a sharp tap near the end of the distillation as a precaution against adhering moisture drops. When no more water can be collected the top surface of the water is adjusted to the zero mark of the burette and the mercury meniscus read. The indicated volume of water is corrected for meniscus errors by the addition of a constant value. The weight of water is then calculated with allowance for temperature. It is advisable to keep the apparatus scrupulously clean and free from grease. No trouble has been encountered with the taper joint between the flask and condenser. The apparatus is set up in duplicate on a special stand not shown in the figure.

Some 14 duplicate runs have been made with this apparatus since the procedure was standardized for routine use. The moistures in these runs varied from 1 to 29 per cent. The differences between duplicates in 11 runs with xylene ranged from zero to 0.14 per cent, with an average of 0.05 per cent. The average difference in three toluene runs was 0.06 per cent. The apparatus and procedure are therefore capable of giving consistent results. The results are reported to the nearest tenth per cent.

Summary

Lower rank coals should be classified by analyses on the moist-coal basis. For this the true moisture of coal in the seam is required, and a rapid, routine procedure for determining this value is described. Tables and curves are given representing the moisture-holding characteristics of a number of Alberta coals.

Methods for partial drying or air-drying of coal are outlined, as well as methods for determining residual moisture. The nature and magnitude of errors in reported values due to oxidation and decomposition are discussed.

Appendixes describe two forms of air-drying apparatus, and a distillation apparatus for moisture determinations.

DISCUSSION

(*A. C. Fieldner presiding*)

H. J. Rose, Pittsburgh, Pa.—Dr. Cady, of the Illinois Geological Survey, asks a number of questions in connection with this paper:

1. Some of the moisture as reported in the as-received form of the proximate analysis consists of moisture derived from the clay minerals in the coal. Is there any noticeable difference in the general character of the curves, such as shown in the figure for low-ash and high-ash coals? In other words, will the relative humidity in the drying oven affect removal of moisture from both the coal and the mineral matter equally?

2. Do the moisture determinations made by method E agree closely with those determined by methods A and D?

3. Is it correct to consider the true moisture content of the coal as the moisture content at 100 per cent humidity, inasmuch as 100 per cent humidity rarely exists? Could not comparisons be made just as well on a 50 or 60 per cent basis of humidity, which possibly would be more nearly the usual condition?

G. H. Ashley, Harrisburg, Pa.—While Dr. Stansfield's suggestion seems to give a definite result, I think Dr. Gauger's work[15] has shown that Stansfield's results may not be altogether satisfactory. For if the capacity of the coal to hold water changes during mining, shipping and so on, it leaves some uncertainty for the man who is compelled to classify coal from a shipping sample; as, for example, in attempting to classify coals coming into this country from Europe, as they are doing today.

A. C. Fieldner, Washington, D. C. (written discussion*).—The papers on moisture in coal by Messrs. Stansfield and Gilbart, and by Dr. Gauger,[15] are valuable contributions in connection with the classification of coal. These investigators have confirmed the growing belief on the part of the Coal Classification Committee that the normal bed moisture should be considered as a part of the coal rather than as an extraneous material. Furthermore, the paper by H. F. Yancey and K. A. Johnson[16] shows that a much better differentiation of the classes of coal is obtained by plotting moist ash-free calorific values than by using moisture-and-ash-free values.

If coal is to be classified on the mine-moisture basis, a reasonably accurate method for determining this moisture is necessary. The present procedures are subject to at least two sources of error. The sample may be contaminated by extraneous moisture due to water running over the face of the coal bed at the point of sampling, or the face may have dried out to some extent. Also, it is impossible to classify a coal on the basis of samples taken from shipments, since these may have changed in moisture content after mining.

Stansfield and Gilbart have described a promising method for estimating the total moisture in coal at 100 per cent humidity. They present comparisons of true and total moisture as usually determined for 21 samples of coal ranging in moisture content from 3.8 to 33 per cent. The maximum deviation of total from true moisture ranged from +0.8 to −0.4 per cent. This deviation is scarcely large enough to justify the extra work involved in the Stansfield method, except where it is necessary to classify a sample which contained surface moisture or had opportunity to lose some of its bed moisture.

It is probable that the moisture in samples from different points in the same mine will vary in moisture content. In order to obtain statistical information on moisture variation, analyses of mine samples from 324 mines in various states were tabulated in four groups according to the total moisture content. The analyses were taken from publications of the U. S. Bureau of Mines. Table 5 gives a summary of the results of this tabulation. The average of the maximum differences for 100 mines ranging from 1 to 5 per cent moisture was 0.8 per cent. The greatest maximum difference was 3.0 per cent; and 95 per cent of the mines showed maximum differences not exceeding 2.0 per cent. The differences were larger for coals containing more than 5 per cent moisture. The average of the maximum differences for 200 mines ranging from 5 to 20 per cent moisture was 1.4 per cent; the greatest maximum difference was 3.8 per cent; and 92 per cent of the mines showed maximum differences of less than 2.5 per cent. Twenty-four mines ranging in moisture from 20 to 50 per cent showed an average of maximum differences of 2.1 per cent and a greatest maximum of 4.2 per cent. In this group 90 per cent of the mines showed less than 3.0 per cent maximum difference.

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* Published by permission of the Director, U. S. Bureau of Mines.

[15] See page 148.

[16] See page 171.

It is evident from this statistical study that in 90 per cent of the cases of coals containing less than 20 per cent moisture, the maximum deviation of the percentage of moisture in samples from the same mine will not exceed 2.5 per cent. A 2.5 per cent deviation in moisture causes a deviation of approximately 375 B.t.u. in a high-rank coal on a moisture-and-ash-free basis. This effect of the moisture variation is approximately 50 per cent greater than the variation in the calorific value of the moisture-and-ash-free coal from the same mine. Hence the variation in total moisture is the limiting factor in reducing the probable error of the analytical values used in the classification of coal on a moist basis.

It is possible that this range in moisture can be reduced in future mine sampling for the purpose of classifying coal, by using especial care to avoid contamination with surface moisture. Some of the large moisture variations of Table 5 may have been due to this cause. A further diminution of the probable error can be made by increasing the number of samples to be averaged. The probable error of the average of four samples is one-half that of one sample; and, finally, if still greater precision is needed, the coal may be rehydrated according to the Stansfield method.

Table 5.—*Differences of Percentage Moisture in Face Samples of Coal from the Same Mines*

(3 to 5 samples from each mine; 100 mines from various states in each group)

Group No.	Total Moisture, Per Cent	Greatest Maximum Difference Occurring Once in 100 Mines, Per Cent	Average of Maximum Differences for 100 Mines, Per Cent	Mines Showing Maximum Differences of Moisture, Percentages Less Than			
				1.0	2.0	2.5	3.0
1	1 to 5	3.0	0.8	80	95		
2	5 to 10	3.8	1.5	27	80	92	96
3	10 to 20	3.4	1.3	37	74	92	96
4[a]	20 to 50	4.2	2.1	8	42	80	90

[a] Only 24 mines in this group.

The xylene distillation method has been used in the Bureau of Mines laboratories on various occasions for the determination of total moisture in coal and other solid fuels. The results obtained by this method usually checked closely with those obtained by the standard Bureau of Mines method,[17] in which the coal is air-dried before the final laboratory sample is prepared. The xylene tests were made according to the method described by P. Schläpfer.[18] Results obtained on comparing the two methods on a series of split samples are given in Table 6. The good agreement obtained between these radically different methods for determining total moisture indicates that both are essentially correct.

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[17] F. M. Stanton and A. C. Fieldner; revised by W. A. Selvig: Methods of Analyzing Coal and Coke. U. S. Bur. Mines *Tech. Paper* 8 (1926) 2–3.

TABLE 6.—*Comparison of Total Moisture Obtained on Split Samples by the Xylene and Bureau of Mines Air-drying Method*

Description of Sample						Time of Air-drying, Hr.	Total Moisture, Per Cent		Difference
Designation	Rank	State or Province	County	Bed	Mine		B. of M. Method	Xylene Method	
	Bituminous	Alberta					6.9	5.9	−1.0
	Subbituminous	Alberta					17.3	17.6	0.3
BE.....	Subbituminous	Wash.	Whatcom	Bellingham	Bellingham	68	8.6	7.0	−1.6
BD....	Subbituminous	Wash.	King	Jones	New Black Diamond	164	12.7	12.5	−0.2
I.......	Subbituminous	Colo.	Boulder	No. 1 Laramie	Industrial	140	19.1	19.5	0.4
B......	Subbituminous	Wash.	Thurston	Bagley	Bucoda	164	21.2	20.6	−0.6
G......	Subbituminous	Colo.	Weld	No. 1 Laramie	Grant	164	25.4	26.2	0.8
R......	Subbituminous	Wash.	Lewis	Reliance	Reliance	164	28.3	28.3	0.0
A49960.	Lignite	N. D.				72	36.5	36.3	−0.2
70968...	Lignite	N. D.	Mercer	Zap	Indian Head	68	33.7	34.8	1.1
TT.....	Lignite	N. D.	Burke	Noonan	Truax Traer	164	37.7	38.7	1.0
						Arithmetical average.....			0.7
						Algebraic average........			0.0

H. C. PORTER, Philadelphia, Pa.—Along the line of what has just been said, I would like to suggest that the variation in coal of the same analysis as to physical structure is sometimes very marked. The anthracite people know that particularly: in the anthracite seams they have hard coal and softer coal. The harder will burn at a slower rate than the softer, which is probably due to the physical structure, to its capillary pore structure, and that is tied up with its moisture-retaining capacity.

That same thing probably holds true with the bituminous seams, bone coal, for instance, and the parts called durain may be found to be of that character. Fusain and other physically different parts of the seam will retain moisture in different degrees. That point should be taken into consideration in sampling the mine.

E. STANSFIELD and K. C. GILBART (written discussion).—Dr. Cady and Dr. Porter point out the probability that the pure coal substance and its mineral impurities will have different moisture-holding capacities, as will also physically different parts of the seam. This is undoubtedly true. We have found some coals in which the moisture increases with increase of ash, more in which it decreases, but many coals in which the moisture appears almost independent of the ash percentage. Accurate sampling is important, as suggested; but for coal classification the coal substance rather than the mineral impurity is under consideration, so that clean samples are desirable if not essential.

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[18] P. Schläpfer: *Ztsch. f. angew. Chem.* (1914) **27**, 52 to 56.

Dr. Cady's question as to whether method E agrees with methods A and D was answered in the affirmative by Mr. Fieldner, who also confirms our view that this agreement between such widely differing methods indicates that both are essentially correct. We found good agreement on the whole, but a slight tendency to higher values with method E when xylene is used. We incline to the belief that toluene would be more generally satisfactory.

Dr. Cady suggests that as 100 per cent humidity rarely exists it might be better to use 50 or 60 per cent humidity values for comparison. Our suggestion is that the 100 per cent humidity value normally represents the coal as it exists in the seam, and we found a very erratic relation between the 100 per cent humidity value and the 60 per cent value, as shown in the last column of Table 4. Many years' experience with values of moisture retained after air-drying to 60 per cent humidity has led us to the belief that this value is not satisfactory for coal classification. We would prefer to use a higher value, such as 96 per cent, if the 100 per cent value is regarded as impracticable.

Dr. Ashley refers to the difficulty of classifying a coal from a shipping sample. It might be designated as having the analysis of such a class of coal. Classification of any sample of unknown history will always be unsatisfactory. It would be interesting to know whether Dr. Ashley would regard a coal as changed in classification because prolonged exposure in a pillar or in a stock pile had changed its analysis.

Mr. Fieldner calls attention to the comparatively small discrepancies between the total moisture found and the "true moisture," and he questions the need for determination of the latter. We only suggest our vapor-pressure method for special determinations with a view to classification or referee work. The variations shown in the table supplied by Mr. Fieldner emphasize the need for such a determination. No mere multiplication of analyses will be of any advantage if there is free water on the coal in part of the mine.

We do not care for the name "rehydration method" suggested by Mr. Fieldner for our determination. Although we claim that the hysteresis error is slight when the sample has only been subjected to a limited amount of drying, we would always endeavor to work with samples that did not require rehydration. The name "vapor-pressure method" might be employed. An essential feature of our work is the prevention of oxidation. We are firmly convinced that failure to prevent oxidation has been a serious source of error in many experiments.

In Appendix I the use of a calcium chloride solution of 1.30 sp. gr. was specified to give a 60 per cent humidity atmosphere. We have recently used a potassium carbonate solution of about 1.495 sp. gr. in the latest apparatus, on account of the corrosive action of the other solution.

Condition of Water in Coals of Various Ranks

By A. W. Gauger,* State College, Pa.

(New York Meeting, February, 1932)

For perhaps one hundred years scientists have been engaged in attempts at devising a satisfactory method of classification of coal. During this time many charts, tables and graphs have been proposed, some of which have had considerable merit, but no one system has been found to be applicable to all ranks of coal, nor has any one system received general acceptance. In many of the proposals, water eliminated between room temperature and 105° C. has been regarded as extraneous matter and calculations have been made to the air-dry or water-free basis. Such methods of calculation do not lend themselves to satisfactory grouping of the high-moisture coals.[1,2] A consideration not only of the properties of the high-moisture coals and their behavior on drying but of the generally accepted ideas of the coalification process and the role of water in this process indicates that water is an essential constituent of all coals and therefore should not be neglected. For example, in such low-rank coals as the lignites of the Northern Great Plains' deposits the inherent moisture of the lignite as mined plays a very important part in the physical properties thereof. Freshly mined lignite contains upwards of 30 per cent water and is extremely tough and resistant to shock. Upon exposure to the atmosphere, the lump loses part of its moisture and checks and splits, leaving a friable and broken up residue. So far as we have been able to determine, water is the only thing lost in this process and we are forced to the conclusion that moisture plays an extremely important part in the structural properties of the original lignite.

With reference to the coalification process, Mack and Hulett[3] pointed out in 1917 the important part that water played therein. They observed that water cannot be removed from peat or coal by pressure alone, but only as the result of chemical reaction, and suggested that peat in its early stages may be largely a hydrosol. Slow chemical changes occur over a long period of time until finally the mass becomes more largely a hydrogel. These views are substantiated to some extent by the experiments of Bergius[4] and Berl,[5] both of whom succeeded in transforming cellulose into substances resembling bituminous coal by heating it under pressure in presence of water.

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[1] Ashley: *Trans.* A. I. M. E. (1920) **63,** 782.

[2] Thom: *Trans.* A. I. M. E. (1925) **71,** 282.

[3] Mack and Hulett: *Amer. Jnl. Sci.* (1917) **43,** 89.

In consequence of considerations such as these, various investigators have reached the conclusion that the water content is an essential constituent of natural coal. If one accepts this view two questions immediately arise: (1) How may the moisture in coal be determined accurately? (2) What is the condition of water in coal?

Accurate Determination of Water in Coal

Many years ago, certainly as early as 1912,[6] attention was drawn to the fact that the analysis of coals for water content involves a number of complicating factors. In 1914 Hinrichsen and Taczak[7] tested the following eight methods of determining water in coal:

1. The sample was dried in an open flat crucible in air in an oven at 105° C. for one hour.
2. The sample was dried in a covered crucible in air in an oven at 105° C. for one hour.
3. The sample was dried in an open flat crucible in an oven filled with CO_2 at 105° C. for one hour.
4. The sample was dried in a crucible set in a beaker kept filled with CO_2 in an oven at 105° C. for one hour.
5. The sample was distilled in toluol.
6. The sample was distilled in xylol.
7. The sample was distilled in cumol.
8. The sample was heated to 100° C. in an open crucible in a vacuum for one hour.

The highest results were obtained by method 8, and the next highest were obtained upon distillation in xylol. Hinrichsen and Taczak laid the higher results obtained by heating in a vacuum to the loss of noncondensible gases and recommended the xylol distillation method. Apparently they made no attempt to distinguish between the water and permanent gases given off in a vacuum.

Mack and Hulett[8] and later, Hulett, Mack and Smyth[9] studied the method of determination of water in four different coals from the United States and found that drying at 105° C. for one hour was insufficient to remove all of the water originally present as such. Their results, which are plotted in Fig. 1, indicate that water of decomposition is not given off by the coals that they tested until a temperature of at least 275° C. is reached. By this method upwards of 30 per cent more water was obtained than by the normal method of water determination. Noncondensible gases are given off, however, even below 105° C. and a replotting of their data for the volume of gas evolved at different temperatures indicates that decomposition of the coal does set in at a lower temperature than that indicated by the break in the moisture-temperature curves. (See Fig. 1.) It must be emphasized that these investigators actually weighed the water given off by the coals.

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[4] Bergius: *Jnl.* Soc. Chem. Ind. (1913) **32,** 462.

[5] Berl: Paper presented before Third International Conference on Bituminous Coal, Carnegie Institute of Technology, Pittsburgh, Pa., November, 1931.

[6] Hillebrand and Badger: *Proc.* 8th Int. Conf. Applied Chem. (1912) **10,** 187.

[7] Hinrichsen and Taczak: *Mitt. k. Material pruefungsamt* (1914) **32,** 290.

[8] Mack and Hulett: *Amer. Jnl. Sci.* (1917) **43,** 89.

[9] Hulett, Mack and Smyth: *Amer. Jnl. Sci.* (1918) **45,** 174.

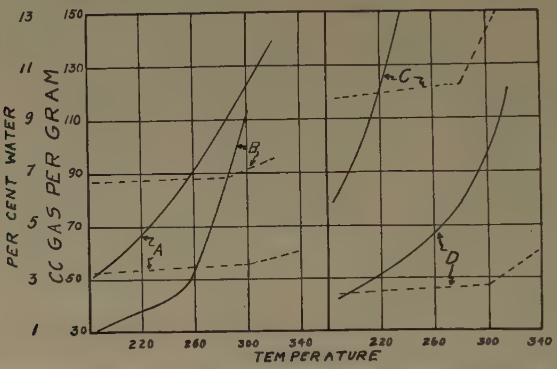

Fig. 1.—Temperature data (Hulett). Broken lines indicate water; solid lines, gas.
A, New River coal; *B*, Indiana coal; *C*, Wyoming coal; *D*, Pittsburgh coal.

Another factor which comes into play in the normal method of determination of moisture is the property that various coals have of sorption of different gases. The work of Graham[10] and others indicates that coal has a tremendous power for sorption of various gases such as hydrogen, oxygen, nitrogen, methane and carbon dioxide. Graham's experiments indicate that the sorptive capacity increases tremendously in dry coal over wet coal. When the coal is dried in air at 105° C., therefore, it may gain weight due to sorption of oxygen and nitrogen or it may lose weight because of loss of moisture or carbon dioxide formed by oxidation. The work at the universities of Minnesota and North Dakota,[11] as well as the work of Parr[12] and of Rosin,[13] indicates that some coals at least may be oxidized with evolution of CO_2 at temperatures much lower than the normal temperature of the drying oven.

To sum up, there are five things that may occur in the present moisture determination, all of which affect the results:

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[10] Graham: *Trans.* Inst. Min. Engrs. (1916–17) **52**, 338.

[11] Eaton, Brady, Gauger, Lavine and Mann: *Ind. & Eng. Chem.* (1931) **23**. 87.

[12] Parr and Coons: *Ind. & Eng. Chem.* (1925) **17**, 118.

[13] Rosin: *Fuel Sci. Prac.* (1929) **8**, 66; see also *Braunkohle* (1928) **27**, 241, 282.

1. Water may be given off.
2. Gases such as CH_4, CO_2 and CO may be lost.
3. The coal may gain weight due to sorption of oxygen or nitrogen.
4. It may lose weight because of loss of water or carbon dioxide formed by oxidation.
5. Some of the moisture may still be retained in the coal.

While we must admit that the present method of determination of moisture in coal may be satisfactory for all practical purposes, it is at the same time obvious that the results obtained by it represent nothing more nor less than loss in weight upon heating under certain conditions rather than water content. The author retains the opinion that for fundamental scientific purposes it is not satisfactory and that it is essential that this whole question of moisture determination be given a thorough fundamental study. We may conclude at this time, first, that the present loss in weight determination does not accurately represent moisture and second, that heating to 105° C. even in a vacuum is insufficient to remove all the moisture from some coals.

Condition of Water in Coal

For the past five years the author has given considerable attention to the study of moisture in lignite. More recently the work has been extended to a consideration of the moisture in coals of other ranks. Although the latter work is by no means completed it is possible at this time to draw some conclusions with reference to the condition of water in coals of various ranks from a consideration of these experiments as well as other published data.

Water recoverable from coal is obtained from the following sources: (1) decomposition of organic molecules (sometimes called "combined water"); (2) surface adsorbed water; (3) capillary condensed water; (4) dissolved water; (5) water of hydration of inorganic constituents of the coal.

Eliminating source 1, since that does not represent water existing as such in the coal, a survey of the experimental data available at the present time indicates that the mechanisms of surface adsorption and capillary condensation play the most important part in the H_2O-coal substance system. In order to obtain an idea of the condition of water in lignite, the author and his collaborators,[14,15] studied the relationship of the moisture content of lignite to the relative humidity of a surrounding air phase. Both static and dynamic methods were tried and the former was found to be more convenient and quite satisfactory for high-moisture

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[14] Lavine and Gauger: *Ind. & Eng. Chem.* (1930) **22**, 1226.

[15] Larian, Lavine, Gauger and Mann: *Ind. & Eng. Chem.* (1930) **22**, 1231.

coals. The general method was to expose samples of raw lignite, pulverized to 80 to 100 mesh, to atmospheres of carefully controlled relative humidities at constant temperatures until a steady state was reached. The relative humidities varied between approximately 0 and 100 per cent and the temperatures were 20°, 40° and 50° C. The samples were weighed from time to time and when successive weighings did not vary by more than ±0.5 mg. it was assumed that a steady state existed. All of the samples were then dried over concentrated sulfuric acid and reexposed to the original relative humidities as before. In this manner both dehydration and hydration curves were obtained.

The moisture contents of the samples at different vapor pressures were then calculated on the basis of the H_2SO_4-dried coal, from the final weights of the samples, both for dehydration and hydration at the different temperatures of experimentation.

This study was later extended to include various woods, peat and coals of other ranks. Through the courtesy of Dr. Irvin Lavine and Mr. Moses Gordon, of the Division of Mines and Mining Experiments, University of North Dakota, the author is able to include in this paper some data for high-rank coals. Although the method used is not the most accurate for low-moisture coals (a new method of study is being worked out for high-rank coals) it is possible, nevertheless, to draw certain conclusions from the results.

Fig. 2 represents graphically the available data for Dakota lignite, Minnesota peat, certain woods, processed lignite and three bituminous coals. It must be emphasized that the zero water content as plotted in the curves does not mean that all of the moisture has been removed from the lignite; it only means that any residual water is either present as "combined water" or as adsorbed water and possesses a negligible vapor pressure. Three samples of lignite from different fields in North Dakota showed an average loss in weight of 1.78 per cent at 110° C. after having been previously dried over sulfuric acid. In order to remove the adsorbed water the temperature must be increased sufficiently to increase the vapor pressure, and in order to remove the combined water the temperature must be raised to the decomposition point. (Compare the results of Hulett and his co-workers referred to previously.)

Comparison of the results obtained with lignite with the work of Van Bemmelen,[16] Zsigmondy,[17] Anderson[18] and others indicates that lignite retains moisture in much the same way that silica gel does. The abnormal vapor pressures as well as the apparent hysteresis may be explained if we assume that lignite consists largely of a colloidal mass with the properties of a nonswelling gel. Such a structure was proposed for German brown coal by Winter[19] in 1913 on the basis of certain optical properties. If we concede that the colloidal groundmass has a spongelike structure with ultramicroscopic capillaries of varying radii, these capillaries may be considered as holding liquid water, the meniscus being concave towards the vapor.

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[16] Van Bemmelen: *Ztsch. anorg. Chem.* (1897) **13,** 233; (1902) **30,** 265.

[17] Zsigmondy: *Ztsch. anorg. Chem.* (1911) **71,** 356; (1912) **75,** 189; (1912) **79,** 202.

[18] Anderson: *Ztsch. physikal. Chem.* (1914) **88,** 191.

A freshly mined lump of lignite loses moisture very rapidly upon exposure to the atmosphere because much of the water is but loosely bound. The vapor pressure of this moisture is equal to that of a plane surface or, in other words, is normal. Upon further loss of moisture, the vapor pressure decreases, which may be explained on the assumption that water is beginning to evaporate from the capillaries. As the capillaries from which water is being removed become continuously smaller and smaller, it may be shown on purely thermodynamic grounds that the vapor pressure of the concave surface must be lowered proportionately. In fact, it is possible by means of the well-known Thompson[20] equation to calculate the radii of the capillary spaces in lignite if we assume that the thickness of the adsorbed layer is not too large a fraction of the capillary diameter. By calculating successively the percentage of water held in the pore radii intervals between that corresponding to 100 per cent relative humidity ($>56.73 \times 10^{-7}$ cm.) and all the other radii an estimate of the relative distribution of the pore sizes is obtained. These calculations have been made for the samples tested and are given in Table 1. Fig. 3 shows the distribution graphically.

From a study of Table 1 certain tentative conclusions may be drawn with reference to the formation and structure of lignite and other coals. In going from wood to peat it is observed that the number of large capillaries increases. This is evidenced by the fact that over 50 per cent of the water in peat is held in capillaries larger than 6.20×10^{-7} cm., whereas only 41.8 per cent of the water in wood is held in capillaries of similar size. On the contrary, however, in comparing peat with lignite and the bituminous coals examined, a progressive decrease in the relative number of capillaries of the larger sizes is observed. When the change in water content during the coalification process is also considered, it becomes evident that there has been an increase in pore volume in going from wood to peat, and a progressive decrease in pore volume from peat to bituminous coal. At the same time, in the transformation of wood to peat there has been a relatively larger increase in number of large capillaries, whereas in the later stages of the coalification process the decrease in number of large capillaries overshadows the decrease in number of small capillaries.

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[19] Winter: *Glückauf* (1913) **49,** 1406; (1914) **50,** 445; *Kolloid Ztsch.* (1916) **19,** 8.

[20] Thompson: *Phil. Mag.* [4] (1871) **42,** 448.

TABLE 1.—*Moisture Held within Various Pore Radii Intervals*

Relative Humidity, Per Cent	Radius, Cm. × 10^{-7}	Bound Water,[a] Cumulative Per Cent						
		Wood	Peat	Lignite	Ala.	Va.	Pgh.	Proc. Lig.
98.0	>56.73	7.8	18.2	14.8	1.6	2.4	3.9	10.2
90.0	>10.31	25.7	44.7	38.5	15.2	16.8	18.3	27.6
84.0	> 6.20	41.8	52.9	44.3	30.4	20.4	29.1	39.3
79.2	> 4.65	46.1	55.9	47.7	34.6	26.8	31.7	42.6
76.0	> 3.93	49.3	58.2	49.4	38.7	30.8	34.6	44.0
66.0	> 2.60	57.9	62.9	54.1	50.8	38.4	38.9	52.3
52.0	> 1.65	68.3	68.5	61.8	59.7	45.6	51.3	59.1
45.0	> 1.35	71.3	70.6	66.6	61.2	47.6	52.6	62.9
33.0	> 0.97	76.5	76.9	75.9	69.1	56.0	61.1	66.8
20.0	> 0.67	78.6	82.9	81.8	80.6	63.6	69.9	75.0
14.9	> 0.57	85.7	87.1	85.9	82.7	73.2	73.5	78.5
9.2	> 0.45	89.5	88.5	89.6	86.9	80.0	81.1	83.3
3.5	> 0.32	95.0	94.0	92.5	94.2	90.8	91.8	95.1
0.0	> 0.00	100.0	100.0	100.0	100.0	100.0	100.0	100.0

[a] Percentage of bound water $= \frac{T - E}{T} \times 100$, where T = moisture at 100 per cent relative humidity, and E = moisture in any other humidity.

During the transformation from wood to peat, biological and chemical decay take place, resulting in the formation of a hydrosol in which the larger portion of the water content is contained in capillaries larger than 6×10^{-7} cm. In other words, the formation of peat from wood is accompanied by swelling and an increase in large capillaries, due in part no doubt to biological causes. As the process goes on with the aid of pressure, the small pores and openings undergo but slight decrease in size. On the other hand, the number of larger openings decreases materially. Water is lost and the mass becomes a hydrogel. As the coalification process proceeds to the coals of higher rank, there is a continual decrease in the number of capillaries of all sizes, but this is more marked with those of the larger sizes. In consequence, the moisture isotherm is shifted more markedly to the left in the region of higher vapor pressure. (See Fig. 2.)

This phenomenon is brought out very distinctly in the effect of partly dehydrating lignite by means of saturated steam at high pressure. In this process[21] lignite is exposed to saturated steam at a pressure of 13 atm. or more for a suitable length of time. The pressure is then released and warm air blown through the mass. When removed the lignite has dried to about 15 per cent water, at the same time retaining its original form-value. Such processed lignite will dry to a lower moisture content than will raw lignite under similar conditions. The curve for processed lignite in Fig. 2 indicates that there has been not only a distinct loss in the number of larger capillaries but that the phenomenon of hysteresis has almost disappeared. In other words, apparently we are dealing with a case of reversible sorption. In this connection the fact must be emphasized that the desorption and sorption graphs do not represent equilibrium conditions when dealing with lower rank coals, consequently it is not strictly accurate to speak of the vapor pressure of a low-rank coal, inasmuch as the same value cannot be reached from both sides.

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[21] Lavine, Gauger and Mann: *Ind. & Eng. Chem.* (1930) **22,** 1347.

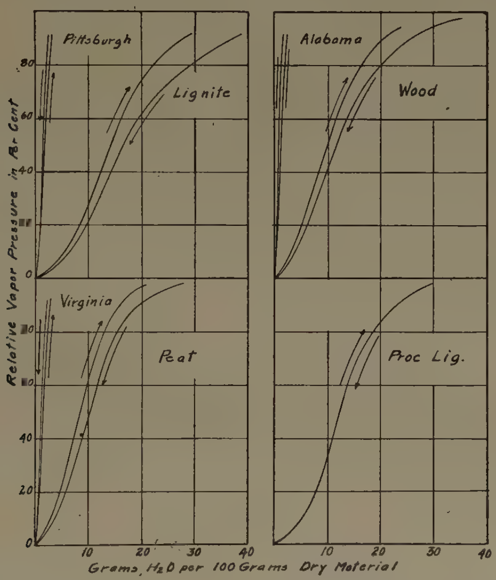

FIG. 2.—AQUEOUS TENSION OF VARIOUS SUBSTANCES.

1. Pittsburgh coal, Champion No. 4, washed coal used in blending. Pittsburgh Coal Co.
2. Roda, Big Stone Gap, Va. Stonega Coal & Coke Co.
3. Mary Lee, unworked coal. Flat Top mine, Jefferson, Alabama.
4. Processed lignite. Represents average of three runs; two at 13 atm. maximum steam pressure on Rupp and Velva lignite; one at 15 atm. maximum steam pressure on Velva lignite. Samples air-dried after processing (average air-drying loss 6.5 per cent).
5. Lignite from Knife River Coal Mining Co., Beulah, N. D.

ANALYSES

	(1)	(2)	(3)	(4)	(5)
Moisture	1.2	1.7	1.2	9.3	35.6
Volatile matter	35.7	36.1	27.9	39.8	27.9
Fixed carbon	58.7	59.5	55.0	47.7	30.9
Ash	4.4	2.7	15.9	5.8	5.6
British thermal units	14,430	14,770	12,600	10,003	7,110

The phenomenon of hysteresis occurs in many cases of sorption. No general explanation is yet available but the author submits that in the case of lignite it can be explained on two assumptions: (1) The shrinking of lignite upon drying causes a collapse of some of the capillaries and the dried material can no longer take up as much water as it held originally; (2) the replacement of moisture on the walls of some of the capillaries by adsorbed gases makes it difficult to wet the capillaries. As a result, the diameter of the surface of the water in the capillary is diminished and the vapor pressure at any given water content decreases accordingly. The behavior of the steam-dried lignite is considered to be good evidence in favor of the collapse of capillary theory. In this process the colloidal mass has been set and further permanent shrinkage does not occur upon complete dehydration.

The coalification process appears to have a similar effect on lignite, for the higher rank coals, if anything, appear to take up slightly more water upon rehydration. In other words, the hydration curve lies outside the dehydration curve. This phase of the problem requires further study by more refined methods, because the larger values could be explained by adsorption of oxygen and nitrogen from the atmosphere. In connection with the high-moisture coals this factor is negligible, but it becomes increasingly important as the moisture content of the coal decreases.

With reference to the comparatively large quantity of water still retained by coals after drying over sulfuric acid at ordinary temperatures, the meager evidence now available indicates that water given off in a vacuum at temperatures up to perhaps 200° C. consists of adsorbed water. At some temperature above that point decomposition sets in, liberating what has been termed chemically "combined" water. It is hardly likely that such water is present as water of hydration of the organic compounds that make up the coal substance. It is more reasonable to assume that the decomposition of compounds containing carboxyl groups results in a splitting off of H_2O. Nor does it seem unreasonable to assume that a small fraction of the water considered (on the basis of vapor-pressure studies) as being condensed in capillaries really consists of surface adsorbed water. If one accepts the Langmuir[22] mechanism for adsorption of gases on solid surfaces as being applicable to the adsorption of water vapor on coal surfaces, the process is a dynamic one and water molecules are condensing on and evaporating off the surface at all times. In consequence of this mechanism, as well as surface changes due to shrinkage, adsorption of gases and other factors, the progressive drying of the coal at the lower relative humidities removes some adsorbed water. Further study is necessary in order to make a more accurate distinction between the three types of mechanism by which water is retained by coal.

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[22] Langmuir: *Jnl.* Amer. Chem. Soc. (1916) **38**, 2221.

Calculations Involving Heats of Combustion

Porter and Ralston[23] showed that heat is produced by wetting dry coal or partly dried coal containing less than its normal percentage of water. To quote from page 16 of their paper, "The relative quantity of heat generated depends on the kind of coal and its relative deficiency in inherent water as referred to its maximum normal content. In other words, the thermal effect of wetting varies directly as some function of the relative vapor-pressure deficiency in the coal." Porter and Ralston

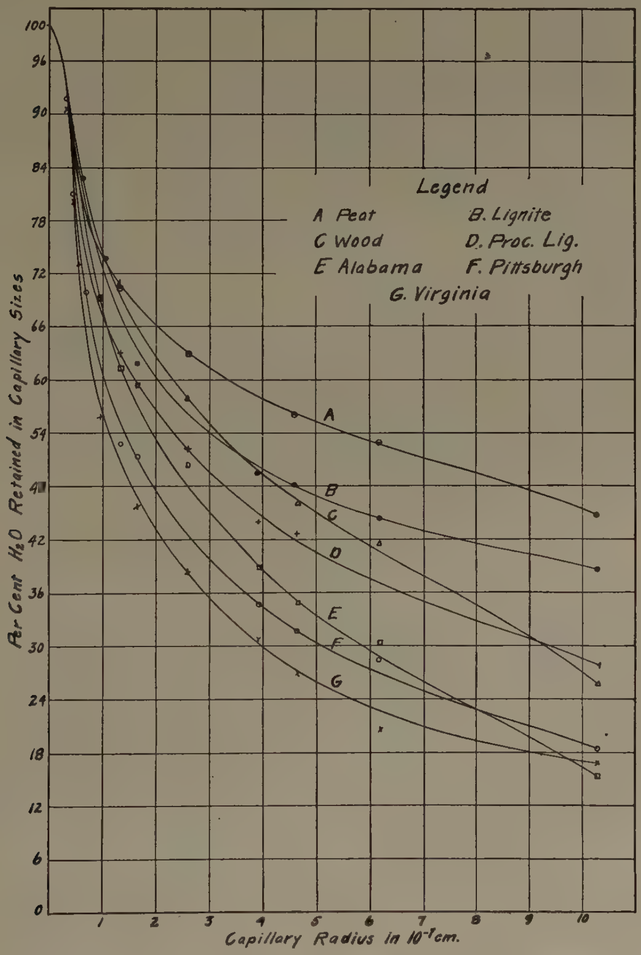

FIG. 3.—DISTRIBUTION OF WATER IN CAPILLARY SIZES.

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[23] Porter and Ralston: U. S Bur. Mines *Tech. Paper* 113 (1916).

consider as "inherent water" all water that does not exhibit normal vapor pressure. In other words, it includes adsorbed water as well as capillary condensed water.

The functional relationship between vapor pressure and heat of vaporization may be obtained from the Clapeyron-Clausius equation

$$\frac{dp}{dt} = \frac{\Delta H}{T \Delta V}$$

in which ΔH is the heat of vaporization of the water, T the absolute temperature, ΔV the change in volume on vaporization and dp the change in vapor pressure, providing one has isotherms at two different temperatures. Lavine and Gauger,[24] assuming that ΔH may be considered a constant and that the ideal gas laws hold under these conditions, calculated the heat of vaporization of capillary water from lignite as between 10 and 40 g. water per 100 g. dry lignite. Their results are reproduced in Table 2.

TABLE 2.—*Heat of Vaporization of Moisture from Lignite*

$$ML = 2.303R \frac{T_1 T_2}{T_2 - T_1} \log_{10} \frac{P_2}{P_1}$$

Where $ML = \Delta H$ = molecular weight multiplied by latent heat of vaporization in calories per gram; R = gas constant; P_1 and P_2 = vapor pressures at absolute temperatures T_1 and T_2 respectively.

$$L = 1164.44 \log_{10} \frac{P_2}{P_1} \times (40 \text{ to } 20)$$

Moisture per 100 Grams Dry Lignite, Grams	Pressure, Cm. Hg.		Heat of Vaporization, Cal. per Gram
	At 20° C.	At 40° C.	
40	1.653	5.355	592.9
35	1.561	5.051	593.8
30	1.433	4.646	594.8
25	1.254	4.099	598.9
20	1.024	3.440	612.8
15	0.737	2.545	626.7
10	0.395	1.411	643.8
			Av. 609.1

Unfortunately, the results do not lend themselves to a ready comparison, but subtracting the normal heat of vaporization of water from the abnormal results for lignite gives a value which should correspond to the heat of wetting. The average increase in heat of vaporization between 10 and 25 g. water per 100 g. dry lignite is about 40 cal. per gram of water. Assuming that the same lignite was used by Porter and Ralston in experiments on both the vapor pressure and heat of wetting, one can calculate a heat of wetting of about 60 cal. per gram of water over the same moisture range. The order of magnitude is the same and the lack of better agreement may be attributed to the previous history of the samples in the two different sets of experiments. It must be emphasized that the heat of adsorption of water vapor on coal surfaces is undoubtedly many times larger than the heat of wetting.

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[24] Lavine and Gauger: *Ind. & Eng. Chem.* (1930) **22,** 1226.

As a direct consequence of this phenomenon, the thermal value of dried lignite will be larger than that calculated from a calorimetric determination made on a wet or partly dried sample. In the latter determination, part of the heat generated is used in overcoming the forces that tend to hold the water in the coal.

Variation of Specific Heat with Moisture Content

Unpublished studies made by Donovan Salley at the University of North Dakota indicate that the specific heat of water in lignite is not normal. In other words, the specific heat of the water is not an additive function for moisture contents between 0 and 10 per cent. Here again the effect of the previous history of the sample plays an important part and makes accurate and constant measurements difficult. The results are in qualitative agreement with those of other investigators but further study is necessary in order to clarify the subject.

Calculation of Moisture to Air-dry Basis

The author feels that there are serious objections to the proposals to calculate all moisture determinations to the air-dry basis. Anyone who has worked with high-moisture coals must realize that moisture percentage on air-dry basis is entirely without significance. The previous history of the coal sample, the time of drying as well as the temperature, relative humidity and air circulation of the drying room all play important parts in the moisture content on the air-dry basis. When lignite, for example, is removed from the coal seam, it immediately begins to lose moisture, and the extent to which it will dry down depends upon the humidity of the drying medium. If, however, this sample is later exposed to an atmosphere of high relative humidity, it will again take up water, but it will not regain quite as much as it held originally. If the sample is then exposed once more to an atmosphere of the original relative humidity, it will again lose water but will not dry down to the moisture content that it reached the first time. This peculiar behavior (hysteresis) may explain some of the anomalous results obtained with low-rank coals upon air-drying. Figs. 4 and 5, obtained by Kempf,[25] clearly illustrate this behavior not only for brown coals but for coals of higher rank as well. The work of Porter and Ralston[26] also shows the hysteresis that occurs when lignite is dried and permitted to reabsorb moisture under controlled humidity conditions, then dried for the second time under similar conditions.

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[25] Kempf: *Mitt. k. Material pruefungsamt* (1920) **37**, 178.

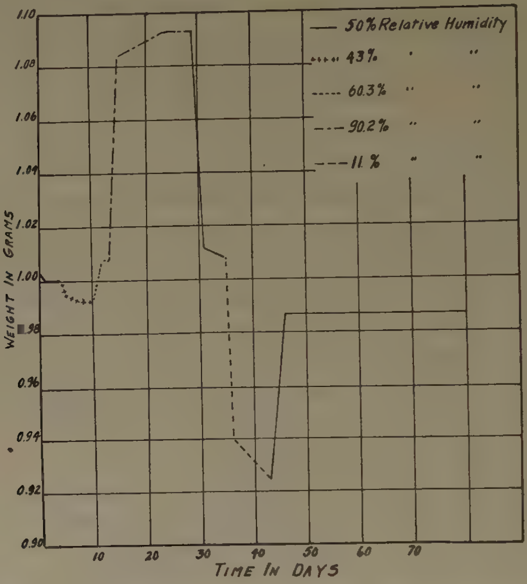

FIG. 4.—BEHAVIOR OF BRAUNKOHLE AT DIFFERENT RELATIVE HUMIDITIES (KEMPF).

Considering the anomalous behavior of the high-moisture coals as illustrated by the experiments described above, it is obvious that the moisture content on the air-dried basis has little significance. It is also evident that the method of sampling is extremely important. In other words, samples must be taken from a fresh face in the seam and must be bottled in sealed containers as quickly as possible. Furthermore, the method of preliminary treatment of the sample in the laboratory requires careful standardization. For example, raw lignite will lose an appreciable amount of moisture during the simple process of grinding.

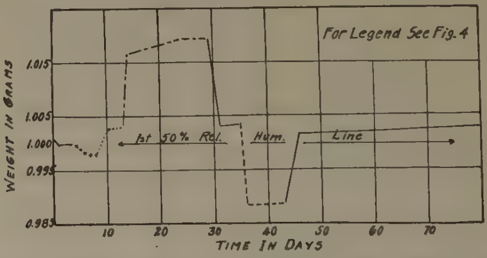

FIG. 5.—BEHAVIOR OF STEINKOHLE AT DIFFERENT RELATIVE HUMIDITIES (KEMPF).

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[26] Porter and Ralston: U. S. Bur. Mines *Tech. Paper* 113 (1916).

SUMMARY

1. The present method of determining the water content of coal is not exact.
2. Water recovered from coal comes chiefly from (1) decomposition of organic molecules, (2) surface adsorbed water, and (3) capillary condensed water.
3. Peat has the properties of a hydrosol. Lignite may be looked upon as a hydrogel with a spongelike structure with ultramicroscopic capillaries of varying radii.
4. The distribution of capillary sizes as calculated from aqueous tension studies indicates a relatively greater progressive decrease in the larger sizes than in the smaller sizes in going from peat to bituminous coal.
5. The evidence at hand favors the acceptance of the view that water is an essential constituent of coal as mined and that drying modifies the original properties of the coal.
6. Abnormal thermal effects of moisture in coal are shown to be directly related to the abnormal aqueous tensions.
7. Further studies by more refined methods are required in order that an accurate understanding may be obtained of the condition of water in coal.

DISCUSSION

(*A. C. Fieldner presiding*)

I. Lavine, Grand Forks, N. D. (written discussion).—Dr. Gauger's paper gives added evidence that the physical structure of coal is a function of its moisture content. The increasing importance of our coals of higher moisture content makes this paper especially valuable at this time.

The relationship between physical structure and moisture content of coal has been brought out strikingly by experiments that were conducted within recent years at the University of North Dakota. In studying various methods for dehydrating lignite without the usual degradation of the lump, it was discovered that lignite could not be dried successfully by the method of controlled humidity. From a theoretical standpoint this method should have proved successful, since it retarded the rate of drying sufficiently to practically equalize the rate of diffusion of the water from the interior to the surface and the rate of surface evaporation. There can be no doubt that the moisture in the coal contributed largely to the failure of this method. In the light of these and other experiments we have concluded that the moisture in lignite must be considered a part of its physical structure and the removal of this moisture by ordinary means cannot be accomplished without affecting its physical structure. Further evidence in favor of this conclusion has been gained from the successful drying of lignite with saturated steam at elevated pressure. In this instance, the physical structure is changed materially before the removal of even a trace of water and dehydration can then take place with a minimum of degradation.

Too little attention has been paid to the properties of the moisture in coal. This is true especially in regard to the lower rank coals, where the high moisture content assumes an important role when methods of economical utilization are being considered. The future development of the vast quantity of low-rank coal in this country will depend largely on a better knowledge of the nature and characteristics of the water in the fuel.

The colloidal structure of coal has been accepted universally without question. It should be emphasized, however, that the term colloidal is general, and indicates that coal possesses properties which are characteristic of a certain class of materials known as "colloidal." It does not indicate the exact nature of the physical structure of the various ranks of coal. Such studies as reported in this paper are but a mere beginning of a field of investigation that promises interesting as well as valuable results.

H. C. Porter, Philadelphia, Pa. (written discussion).—Dr. Gauger has done a good service in calling attention to the importance of characteristic or distinctive water content of coals as an element in their classification. He has emphasized also the admitted and previously known fact that our standardized method for moisture determination is empirical, and gives no fundamentally true information as to the content of water and its condition in the coal. He presents experimental data, mainly on lignites, showing subnormal vapor pressure of all but the free, superficial water in coal, and explains this on the theory of a colloidal hydrogel of capillary pore structure. All of this, both fact and theory, has been brought out by previous investigators, but its importance, for the work of coal classification, justifies its repetition.

It is difficult, however, for the careful reader of this paper to determine the proposals that the author would make for improving our practice in analytical procedure for distinguishing characteristic water content from free accidental moisture, or for incorporating his findings in a method for coal classification. What he has told us as to the condition of the water in coal is generally accepted as approximating the probable actuality, but the problem is to apply this knowledge to practical uses in evolving a more satisfactory method for actual moisture determination and in classifying coals, possibly through use of their normal mine-moisture content as one factor.

These things Stansfield and Gilbart take concrete steps to accomplish,[27] and their proposals, as based on data presented, carry considerable weight. The use of what they call "true moisture" in the coal as a basis of classification, especially of the lower rank coals, is reasonable, since the physical structure and use characteristics of the coal, even of many bituminous ranks, is greatly affected by the presence of such moisture. Neither paper, however, presents a scheme of classification, although Stansfield and Gilbart suggest a classification by analyses (including "true moisture").

To this commentator, it would appear that a better scheme would be to use in classification the "true moisture" as a single characteristic on which to base a distinction between ranks, and use the other elements in analysis on the dry basis for a supplementary classification within groups.

In 1913, G. B. Taylor and H. C. Porter, in a paper on The Specific Heat of Coal and Its Relation to the Presence of Combined Water in the Coal Substance,[28] called attention to the fact that determinations of specific heat on dried and undried coals showed that "the molecular heat of the water in coal is evidently lower than that of free uncombined water and approaches that of the combined water of some crystallized salts. The conclusion, therefore, follows in agreement with that drawn from the vapor pressure relationships that the moisture of coal is in part *physically* or chemically combined . . . The vapor pressure of the moisture in coal does not reach the normal tension of water vapor until the amount present exceeds a certain percentage; up to that percentage, therefore, the water present in coal does not exist as free superficial moisture."

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[27] See page 125.

[28] *Jnl. Ind. & Eng. Chem.* (1913) **5**, 289.

Although this early work on specific heats was not exact, owing to a necessarily approximate correction required for the heat of wetting (either by use of water or of toluene in the experimental procedure), nevertheless the method was of sufficient accuracy to show beyond question a specific heat below the normal additive value. The molecular heat of the 8.4 per cent water present in an Illinois coal was found to be 15.3, and that of 21.4 per cent moisture in a Wyoming subbituminous was 12.0, showing the greater departure from normal in the Wyoming type. Dr. Gauger mentions the matter of specific heat studies on lignite, but appears to have overlooked this early work of Taylor and Porter.

In 1916, H. C. Porter and O. C. Ralston[29] discussed fully the vapor-pressure relationships of different kinds of coal containing different percentages of moisture, and presented theoretical considerations explaining these facts, which recall many of those used by the author of the present paper. For example, in this early publication we find the following: "A peculiar physical condition, possibly a densification, is brought about in the interfacial layer between the solid and liquid phases, tending to lower the normal vapor pressure of the liquid, to increase its latent heat of vaporization, and to liberate a heat of condensation or a heat of formation when the liquid enters into this condition . . . Each particle of the dried coal may be assumed to act as a sponge or a tissue permeated with microscopic pores . . . Theories involving the conception of adsorption or densification of the liquid on the walls of capillary openings or pores have been proposed" (by previous investigators on related types of material) "and studies have been made of hydration phenomena in connection with starch, gelatine, wood fibre and various inorganic colloids."

On the question of an established method of analysis for moisture content, the 1913 paper by Porter and Taylor refers to the difficulties therein involved on account of the different forms of water present and states that "the 'free' water cannot be separated with exactness from the 'combined' water by chemical analysis. Under all ordinary drying conditions, there is a tendency for more than the 'free' water to leave the coal . . . The inherent water is held more tenaciously and cannot be entirely removed by drying the coal in ordinary air containing moisture."

Unfortunately the term "combined water" as used in this early paper was a misnomer, unless it could be understood as water held by physical forces in a state altered as to vapor pressure, specific heat, and perhaps other properties. It was not meant to connote chemical combination.

An analytical method that will show more truly than does the present standard method the actual water content, both inherent (or capillary-condensed) and the free water of normal vapor pressure, is desirable, especially for low-rank coals. Stansfield and Gilbart have proposed the method using distillation with xylene or toluene and their results with it seem to show many points in its favor.

For classification purposes, I believe the relative degree of coalification as shown by ability to reabsorb moisture under uniform humidity conditions will prove alone a sufficient basis for a broad separation of groups according to rank. Within each group, however, other factors, analytical on the moisture-free basis, and physical such as softening characteristics in the coking bituminous group, will be found useful and even necessary.

H. H. Storch, Pittsburgh, Pa. (written discussion).—Dr. Gauger has given a clear presentation of the relevant scientific facts as to the state of the water in coals of various ranks. It is perhaps undesirable at present to place too much emphasis on the detailed calculations of pore sizes in the various materials from peat and wood to anthracite. These calculations lead in most cases to sizes between 10 and 100 times the commonly accepted values for the diameter of gaseous molecules—that is, 10^{-8} cm. Since it is very probable that the "molecules" of peat, wood, or coal are considerably larger than gas molecules (perhaps 1000 times as large), it is evident that we are dealing here with capillaries of molecular dimensions; hence it is very difficult to draw any distinction between the idea of capillary sorption and that of solid solution.

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[29] U. S. Bur. Mines *Tech. Paper* 113 (1916).

The phenomenon of hysteresis might be just as well visualized as an alteration in the space groupings of the molecules upon removal of water. Thus when the chemist dries alumina or silica gel, and finds that he cannot rehydrate them readily to their original moisture content, he does not postulate the collapse of capillaries but rather a rearrangement of groups of molecules combined with the actual breaking of bonds between hydroxyl groups. In order to obtain further information on the state of water in coal and related materials, it might be desirable to study the absorption and removal of other vapors such as alcohols. If, for example, it were found that a coal dried over sulfuric acid at room temperature and subsequently exposed to alcohol vapor did not exhibit the hysteresis phenomena such as are obtained with water vapor, some conclusions concerning the existence of unstable hydrates might be drawn. If hysteresis phenomena were observed with alcohol vapor this datum might be interpreted as indicating the reality of the capillary hypothesis. It would, however, be desirable to experiment with several vapors of widely different chemical composition before much confidence is placed in such conclusions.

A. W. GAUGER.—I think that the key to the solution of the scientific classification of coal lies in more complete knowledge of the constitution of coal.

I will grant with Dr. Porter and with Dr. Stansfield that what we need to know is the true moisture, but I will not grant that the method proposed by the latter will give it to us. I think the experiments of Professor Hulett, wherein he obtained as much as 30 per cent additional water by heating to somewhat higher temperatures, indicate that methods that use only 110° will not give the total water content of the coal.

I was familiar with the pioneering work of Taylor and Porter on the specific heat of coal and its relation to the presence of combined water in the coal substance. However, experience with measurements of specific heat of lignites indicated to me that the problem of its relationship to the water content is exceedingly complex. It seemed inadvisable, therefore, to go into greater detail on this point in the present paper.

With reference to the comments of Dr. Storch, I will agree that the calculation of pore size should be considered as tentative and subject to revision if and when more information becomes available. However, I do not see that anything is to be gained by postulation of rearrangement of molecules instead of collapse of capillaries. The latter, in fact, no doubt will be accompanied by such rearrangements. As for removal of water by actual breaking of bonds between hydroxyl groups, I believe that this should occur at definite vapor pressures. In such event the curves should possess distinct breaks instead of being smooth.

Determination of the Alkali-soluble Ulmins in Coal

By Edgar Stansfield* and K. C. Gilbart* Edmonton, Alberta

(New York Meeting, February, 1932)

When plants decay in a peat bog the woody parts form a brown pasty mass, or peat muck, largely soluble in alkalis. This brown matter has been termed "ulmin." The same material, but commonly black in color, is found in coal, and there is evidence of the similarity between the material forming the main bulk of the coal and the bulk of the more recently decayed material in peat bogs.

The more recent or low-rank coals, such as lignite, little removed from the peat stage, contain notable percentages of alkali-soluble ulmins, but the percentage decreases with increased degree of coalification and is negligible in bituminous or higher rank coals. The determination of the alkali-soluble ulmins in any coal gives, therefore, an indication of its degree of coalification or rank.

Coals which contain few or no ulmins soluble in alkalis do contain ulmins which can be rendered soluble by oxidation. The higher the rank of the coal, the more difficult it is to oxidize the insoluble ulmins and make them soluble. High-rank coals can be differentiated by oxidation tests on the insoluble ulmins; but low-rank coals can be characterized by both their soluble and insoluble ulmins. This paper deals with the determination and significance of the alkali-soluble ulmins present as such in the coal. The chemical nature and composition of ulmins are not discussed.

REVIEW OF LITERATURE

The history of the study of ulmin is well given by Stopes and Wheeler.[1] The action of potash on coal has long been known, as Braconnot in 1819 distinguished between lignites and bituminous coals by stating that the former gave colored solutions with potash. Later the intensity of color was used to give a fuller differentiation. Hoffman[2] distinguished between lignites and lignitic coals by the darker color given by the former. Nicolls[3] used 12 standard color solutions and matched with these the colors given in a closely standardized test. Later workers have endeavored to make the test more strictly quantitative; Charpy and Decorps[4] boiled the coal with 50 per cent aqueous sodium hydroxide and titrated the acidified extract with potassium permanganate in the cold. Francis and Wheeler,[5] in their studies of the oxidizability of coal, boiled the oxidized coal with potash solution, precipitated the extracted ulmins with acid and weighed.

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* Fuels Division, Research Council of Alberta.

[1] M. C. Stopes and R. V. Wheeler: *Fuels in Sci. and Prac.* (1924) **3**, 328.

[2] G. C. Hoffman: Coals and Lignites of the North West Territory. Report of Geol. Survey of Canada (1882–84).

EXPERIMENTAL STUDIES

Studies in these laboratories showed that it was difficult to approach an end point in the extraction with aqueous alkalis; that some coals were hard to wet with the solution; and that the titration value with permanganate depended on the duration and temperature of treatment and strength of solution. The preparation of the sample was also found to be important.

The following procedure was ultimately adopted and has given consistent results with reasonably good checks between duplicates. Freshly crushed coal is air-dried in an atmosphere of neutral gas and the coal then ground, in gas, in a ball mill to just pass a 200-mesh screen. To avoid excessive pulverization the coal is removed from the ball mill from time to time, screened, and only the oversize returned. A 5-gram length of C. P. stick potash is taken and a test tube of a diameter that will just admit the stick. A small length of the potash is placed at the bottom of the test tube, then ½ g. of the pulverized coal is added, and the remainder of the potash is placed on the coal. The test tube is closed with a cork carrying inlet and outlet tubes for gas, and the charge then heated for 20 min., with exclusion of air, in a vertical electric oven maintained at 225° C. The charge soon fuses and the tube is gently shaken to insure good mixing; before the end of the 20 min., however, the charge again solidifies. After cooling, the fused mass is treated with 50 c.c. of distilled water and boiled gently for 15 min. The cooled product is made up to 100 c.c. with distilled water, centrifuged to throw down the undissolved coal, and the solution decanted through a hardened filter paper. The first running of the filtrate is discarded. The ulmins in the collected filtrate are then estimated by the two following methods.

Precipitation of Ulmins.—The ulmins in a 50-c.c. portion of the extract are precipitated by addition of 20 c.c. of concentrated hydrochloric acid. The vessel is centrifuged, the clear liquid decanted off, and the precipitate washed once by decantation with dilute hydrochloric acid. The ulmins tend to go into colloidal solution in water and it is difficult, without loss of ulmins, to completely remove, even with acid washings, the entrapped potassium chloride. The precipitate is therefore washed into a crucible, slowly taken to dryness at 110° C., in absence of air, and weighed. The mixture of ulmins and chloride is then ignited at 650° C. and the loss in weight is assumed to represent the weight of ulmins.

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[3] J. H. H. Nicolls: Summary Report of Mines Branch, Dept. of Mines, Canada (1916) 65.

[4] G. Charpy and G. Decorps: *Compt. rend.* (1921) **173**, 807.

[5] W. Francis and R. V. Wheeler: *Jnl.* Chem. Soc. (1928) 2974.

Titration of Ulmins.—The volume of extract taken for this test is varied according to its strength; that is, its color. If the extract is not darker than standard color[6] No. 5, a 25-c.c. portion is taken. If the solution is darker, it is quantitatively diluted to about that strength, and 25 c.c. of the diluted solution is taken. This solution is boiled gently for 1 hr., with 25 c.c. of 0.1N potassium permanganate solution, in a flask with a reflux condenser. The solution is cooled quickly, made acid with 20 c.c. of 2N sulfuric acid, and 25 c.c. of 0.1N ferrous ammonium sulfate solution added. The excess of ferrous iron is then back-titrated with 0.1N permanganate solution. The result is calculated, following the method of Charpy and Decorps, as the grams of oxygen required to oxidize the soluble ulmins in 1 kg. of coal; this they called the oxidation index.

Duplicate tests are made of the whole determination, including the original fusion. Blanks are run from time to time and the necessary corrections made. The oxidation index is of the order of 10 times the percentage of soluble ulmins by precipitation.

The precautions prescribed in the preparation of the sample for test are necessitated by the susceptibility of coal to oxidation. Francis and Wheeler, Charpy and Decorps, Rose and Sebastian[7] and others have shown that insoluble ulmins in coal are rendered soluble by oxidation. This fact has been confirmed in this laboratory, but it has also been found that the first effect when coal is heated, in air or gas, is a reduction in the percentage of soluble ulmins. This reduction is easily overlooked on account of the rapid increase which follows when the coal is heated in air. It has been observed repeatedly, however, and with many types of coal. Prolonged heating in gas gives somewhat erratic results suggestive of a slow decomposition. The results obtained in two series of tests on Lethbridge coal are shown in Table 1, the minimum values occurred after heating 1 hr. This subject requires further study.

SOLUBLE ULMINS AND CLASSIFICATION OF COAL

The application of determined values of soluble ulmins to the classification of coal can be studied by tests made in 1930 on a series of 13 coals, of widely varying rank, from southern Alberta. Alberta coals are Cretaceous but occur in three geological horizons, Kootenay, Belly River and Edmonton; the latter, the youngest. The coals are thus all comparatively recent but, owing to pressure during the formation of the Rocky Mountains, some of them have been metamorphosed to high rank (anthracite). The rank of the coals, as judged by ordinary analyses, varies with their distance east from the mountain face. The ulmin test,

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[6] J. H. H. Nicolls: Summary Report of Mines Branch, Dept. of Mines, Canada (1916) 65.

[7] H. J. Rose and J. J. S. Sebastian: Changes in Properties of Coking Coals Due to Moderate Oxidation during Storage. *Trans.* A. I. M. E., Coal Div. (1930) 556.

TABLE 1.—*Effect of Heat and Oxidation on the Soluble Ulmins in Coal.*[a]

Duration of Heating at 105° C., Hr.	Heated in Methane		Heated in Air	
	Soluble Ulmins, Per Cent	Oxidation Index	Soluble Ulmins, Per Cent	Oxidation Index
0	6.9	56	6.9	56
1	4.8	35	5.0	40
2	7.8	55	7.7	59
6			8.4	64
24	5.9	68	14.4	133
72	5.3	74	15.4	197
160			23.6	259
600			46.6	531
2712			45.3	470

[a] All results are based on weight of original sample. The coal tested is a strong, black coal of good storage quality. It contains about 10 per cent moisture as mined, has a fuel ratio of 1.25, and a calorific value, with 9 per cent of ash, of about 11,000 B.t.u.

TABLE 2.—*Comparison of Soluble Ulmin and Other Values for Thirteen Alberta Coals*

Sample	Horizon	Distance East of Mountains, Miles	Moisture as Mined, Per Cent	Analysis, Pure Moist Coal		Analysis, Pure Dry Coal		Soluble Ulmins, Pure Moist Coal	
				Fixed Carbon, Per Cent	Calorific Value B.t.u. per Lb.	Carbon, Per Cent	Hydrogen, Per Cent	Per Cent	Oxidation Index
A	Kootenay...		2.2	87.5	15,380	92.45	4.00	0.05	6.6
B	Kootenay...		1.4	84.0	15,560	90.65	4.35	0.0	3.3
C	Kootenay...		1.4	68.2	15,380	88.3	5.15	0.0	0.6
D	Kootenay...		1.2	62.4	15,490	87.25	5.6	0.02	1.5
E	Kootenay...	4	3.3	58.7	14,610	84.7	5.55	0.15	8.5
F	Belly River	6	4.9	54.2	14,240	83.3	6.00	1.8	21.6
G	Belly River	60	9.8	52.0	12,580	79.1	5.35	10.3	146
H	Edmonton..	60	12.9	51.2	11,880	78.65	5.30	40.4	311
J	Belly River	92	15.0	48.9	11,320	77.4	5.15	11.4	129
K	Belly River	110	18.8	45.5	10,370	76.7	5.10	31.0	315
L	Belly River	135	22.6	43.4	9,940	74.9	5.20	37.4	265
M	Belly River	155	26.6	41.3	9,250	73.1	5.15	39.0	232
N	Edmonton..	157	30.3	36.2	8,420	71.65	5.15	40.9	407

on the other hand, emphasizes the geological age, as is shown by the high values given by the two Edmonton horizon coals. The soluble ulmin and other values for these 13 coals are given in Table 2. There is little differentiation between anthracitic and bituminous coals by the soluble ulmins, but the latter differentiate the lower rank coals far more strikingly than do the ordinary chemical analyses. The true significance of these values has not yet been ascertained.

SUMMARY

A procedure is described for determining the alkali-soluble ulmins in coal. The coal is fused with solid caustic potash with rigid exclusion of air, and the extracted ulmins estimated either by precipitation and weighing or by titration with potassium permanganate. The relation of the results obtained to the classification of coal is discussed.

DISCUSSION

(*A. C. Fieldner presiding*)

H. J. Rose, Pittsburgh, Pa.—I have just been handed some questions from Dr. G. H. Cady, of the Illinois Geological Survey. Dr. Cady comments as follows: Illinois coals seem to correspond in general composition to coals G, H, J and K. Do these Alberta coals contain soluble ulmins extractable without previous oxidation?

The answer to that question evidently is yes. According to the paper by Stansfield and Gilbart, these coals were carefully protected from oxidation, and the coals referred to by Dr. Cady yielded from 10 to 40 per cent extractable ulmins. Dr. Cady points out that in that respect they would differ from Illinois coals, which he finds contain no extractable ulmins.

His next point is: What significance is to be attached to the notable difference in coals A to F, the higher rank, as compared with coals G to N, in the amount of soluble ulmins, particularly in the difference between coals F and G, where soluble ulmins suddenly jump from 2 to 10 per cent? Dr. Cady asks, Does this mark a division between different ranks of coal?

In the paper it is said that the true significance of these values has not been ascertained. In other words, this is a progress report and the authors are not ready to draw definite conclusions of the utility of these data in coal classification.

W. T. Thom, Jr., Princeton, N. J.—In this paper we have a very interesting though indirect suggestion regarding coal classification. Europeans have been accustomed to regard geologic age as a major basis for coal classification, whereas American geologists, for the most part, have regarded geologic age as unimportant and metamorphism as of principal importance. That there is evidence supporting the European view seems to be shown by the figures given in Table 2 of Stansfield and Gilbart's paper. Samples H and N, plus the suggestions made by Dr. Cady regarding Illinois coals, appear to show that progressive chemical transformations in coals do make geologic age a factor which cannot be neglected in a scientific classification of coals. In my own studies, I am finding indications that age must be considered, and if tests such as Stansfield and Gilbart have made can be multiplied, it appears that a "soluble ulmin" test can be used, which will indicate significant differences in composition and properties between coals of unlike age, but of similar proximate analysis. From Dr. Cady's remarks, I understand this is also his opinion and expectation.

E. STANSFIELD and K. C. GILBART (written discussion).—The speakers have referred to the significance of the high percentage of soluble ulmins in coal H (Table 2). This appeared to be related to the geological age of the coal; but subsequent work on a series of coal samples taken from west to east farther north in the province did not show a similar abnormally high value for the more recent coals. Two Kootenay horizon coals in the mountains showed no soluble ulmins; two Belly River horizon coals in the foothills showed 1.8 and 2.6 per cent, while 14 Edmonton horizon coals from the plains showed from 9.8 to 31.6 per cent. The ulmin variations in the latter coals showed little relation to the ordinary analyses and no relation has yet been ascertained to other properties of the coals.

It should be noted that the method of fusion with potash employed in this work gives a higher yield of soluble ulmins than the method of boiling with an aqueous solution of potash employed by many workers.

Physical and Chemical Properties of Coal in Relation to Classification*

By H. F. Yancey† and K. A. Johnson,‡ Seattle, Wash.

(New York Meeting, February, 1932)

Physical properties have been used for a long time in characterizing different kinds of coal, and such physical properties as friability and slacking have been included along with chemical properties in grouping coals according to the widely used system of classification adopted by the U. S. Geological Survey. Nevertheless, the usefulness of physical properties for classifying the different ranks or kinds of coal has been limited because it has not been possible to express these properties in definite form as numerical values. Because of the present wide interest in the formulation and adoption of a satisfactory system of coal classification, the Northwest Experiment Station of the U. S. Bureau of Mines, in cooperation with the College of Mines of the University of Washington, has recently completed a study of the physical and chemical properties of approximately 100 different coals ranging in rank from lignite to anthracite.

Purpose and Scope

For this investigation samples were collected in approximately 45 mines and prospects in Washington and 23 mines in other states. The following studies, in addition to proximate and ultimate analyses, were made of the samples: Determination of (1) friability, (2) slacking or weathering characteristics, (3) yield of products by low-temperature carbonization in a laboratory retort, and (4) agglutinating value. Most of the coals examined were subjected to each of the tests enumerated, but with coal of some ranks this was unnecessary in some instances, and in others it was not feasible to do the work involved.

A report which includes the analyses of the coals and the details of the methods of examination used in the investigation is being published by the Bureau of Mines.[1] It is the purpose of the present report to show the results of the friability, slacking, low-temperature and agglutinating studies in relation to several schemes of coal classification, in order to determine which scheme gives the best correlation of the properties enumerated.

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* Published by permission of the Director, U. S. Bureau of Mines.

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[1] H. F. Yancey, K. A. Johnson and W. A. Selvig: Friability, Slacking Characteristics, Low-Temperature Carbonization Assay, and Agglutinating Value of Washington and Other Coals. U. S. Bur. Mines *Tech. Paper* 512.

The samples collected in Washington represented all mines in the state and varied in rank from subbituminous to semibituminous; those taken in other states included samples of lignite, bituminous, semibituminous and anthracite collected from mines operating in the following beds: From three mines operating in the Gamma, Wharton, Buck Mountain, Orchard, Primrose and Mammoth beds of the Eastern Middle field of the anthracite region of Pennsylvania; from one mine of the Thick Freeport and one mine of the Pittsburgh beds of Pennsylvania; from one mine of the Sewell and five mines of the Pocahontas No. 3 beds of West Virginia; from one mine of the Davis bed of Maryland; from one mine of the Highsplint and one mine of the Elkhorn beds of Kentucky; from one mine of the No. 6 bed of Illinois; from four mines of North Dakota; and from three mines operating in beds Nos. 1, 2 and 3 of Wyoming. In addition, a number of samples were collected in states other than those named, on which slacking characteristics only were determined. With one exception (the Vermillion County, Illinois, sample shown in the friability charts), the samples were collected and sealed in air-tight cans in the mine. The moisture content shown represents, therefore, that normal to the bed from which the particular sample was taken.

Method of Presenting Results

The nomenclature of the scheme of progressive classification adopted by the U. S. Geological Survey[2] was used to designate the rank of the coals. Of the various ranks recognized in the Survey's classification, the following are represented in this investigation: lignite, subbituminous, bituminous, semibituminous and anthracite. For convenience, the subbituminous rank was subdivided into two groups, A and B; coals containing less than 20 per cent bed moisture were placed in the A group, and those with more than 20 per cent moisture were placed in the B group. The graphic method of presenting the results of the investigation is used throughout this report. The relationship between physical and chemical properties and rank are shown both on the as-received, ash-free and the moisture-and-ash-free bases, inasmuch as both schemes are favored by various investigators.

Moisture and fixed carbon of the proximate analyses; carbon, hydrogen and oxygen of the ultimate analyses; and British thermal units are used as indexes of rank in these graphs. The points are plotted on graphs of two different forms. The ultimate analyses are plotted on trilinear diagrams according to the plan of Ralston[3] with the carbon plotted along the horizontal axis. Directly below and attached to the trilinear diagram is a rectilinear diagram with some one of the physical or chemical properties plotted along the y axis, the points for which are located directly below the corresponding points of the trilinear diagram. This system is, in effect, the same as plotting the particular physical or chemical property against total carbon as an index of rank. The ultimate analyses have been calculated on two bases: (1) as received, ash-free, sulfur-free and nitrogen-free; (2) moisture, ash, sulfur and nitrogen-free. The first, or as-received basis, includes the moisture normal to the coal as a part of the hydrogen and oxygen of the ultimate analysis. In the second or moisture-free basis, the carbon, hydrogen and oxygen of the dry or moisture-free coal is considered as 100 per cent. In the rectilinear graphs showing the proximate analyses, moisture and fixed carbon, and also British thermal units, are plotted along the x axis and the physical or chemical property is plotted along the y axis.

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[2] M. R. Campbell: The Coal Fields of the United States. U. S. Geol. Survey *Prof. Paper* 100-A (1917) 3–9; American Coal Supply—Its Quantity, Quality, and Distribution. *Proc.* Internat. Conf. on Bituminous Coal. Carnegie Inst. of Tech. (1926) 5–64.

CLASSIFICATION OF COALS EXAMINED

Figs. 1 and 2, arranged in a form suggested by Parr's[4] classification but using fixed carbon instead of volatile matter as an index, show graphically the range in rank of the coals that were studied. The rank of the coals shown in these charts is, with a few exceptions, that assigned to it by the collector of the sample. The heating values in British thermal units are plotted against the percentages of fixed carbon on the as-received, ash-free basis and the moisture-and-ash-free basis. Fixed carbon, as pointed out by Campbell,[5] Fieldner,[6] Ashley[7] and others, was used instead of volatile matter as a ranking index because it increases more uniformly from low-rank to high-rank coal. The disposition of points representing the various ranks of coal are, in general, similar in both charts, but, as is obvious, the inclusion of the bed moisture serves to spread the points in Fig. 1 over a much larger area.

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[3] O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals: U. S. Bur. Mines. *Tech. Paper* 93 (1915). The trilinear method used by Ralston is, in effect, the same as the rectilinear carbon-hydrogen chart of C. A. Seyler which was published in the *Proceedings* of the South Wales Institute of Engineers in April, 1900 and January, 1901.

[4] S. W. Parr: The Classification of Coal. *Jnl. Ind. & Eng. Chem.* (1922) **14**, 919; Univ. of Ill. Eng. Sta. *Bull.* 180 (1928).

[5] Reference of footnote 2.

[6] A. C. Fieldner: The Classification of North American Coals. U. S. Bur. Mines *Inf. Circ.* 6094 (1929) 7; also *Proc.* Second Internat. Conf. on Bituminous Coal, Carnegie Inst. of Tech. (1928) **1**, 640.

[7] G. H. Ashley: Coal Classification; A Review and Forecast. *Trans.* A. I. M. E., Coal Div. (1930) 514.

When bed moisture is excluded, as in the moisture-and-ash-free plot of Fig. 2, the points are compressed in a disproportionate manner into a much smaller area; the low-moisture, high-rank coals are moved only slightly, but the points representing high-moisture, low-rank coals are necessarily shifted to a much greater extent. There is overlapping of coals of adjacent ranks in both cases, but this is much more pronounced in the chart plotted on the moisture-and-ash-free basis. Both charts are drawn to the same scale, but it is apparent that increasing the scale of Fig. 2 would not make it the equivalent of Fig. 1. The different ranks are distinctly separated over the area of the chart plotted on the as-received, ash-free basis, with the exception of the bituminous and subbituminous A

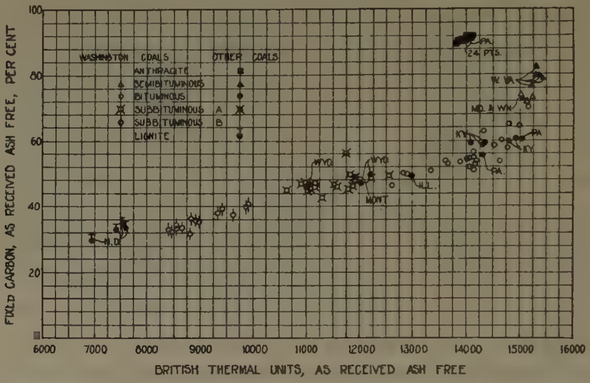

FIG. 1.—CLASSIFICATION OF WASHINGTON AND OTHER COALS EXAMINED IN THIS INVESTIGATION, AS-RECEIVED, ASH-FREE BASIS.

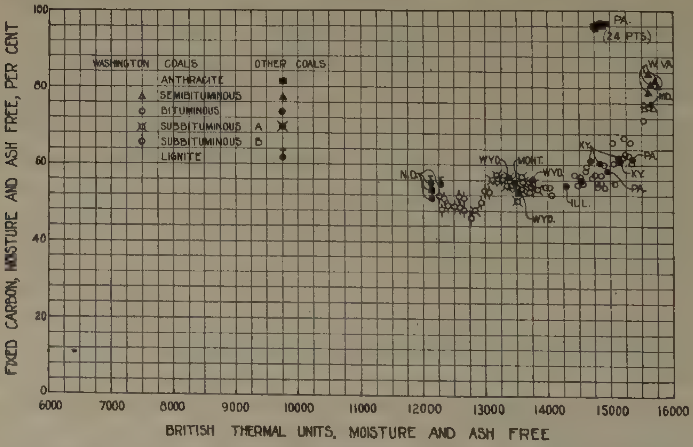

FIG. 2.—CLASSIFICATION OF WASHINGTON AND OTHER COALS EXAMINED IN THIS INVESTIGATION, MOISTURE-AND-ASH-FREE BASIS.

ranks, between which there are a few overlapping coals of questionable rank.

Many American investigators agree that the bed moisture content normally present in all coals is a characteristic and inherent part of the complex substance known as coal, and that, as such, it should be used in any system of classification. Moreover, it is obvious that a classification which recognizes bed moisture represents the commercial product as marketed more nearly than does one which excludes bed moisture. Ash, on the other hand, is a much more variable constituent of coal than bed moisture. In a scheme of classification, therefore, ash may be regarded largely as adventitious, and hence neglected. From a consideration of these facts, it is believed that a classification including bed moisture but excluding ash, as in Fig. 1, portrays more satisfactorily the successive changes to which coal-forming material has been subjected, and the physical and chemical properties of the whole range of products resulting from these changes, than does a classification based upon coal calculated free of both moisture and ash.

It may be argued with obvious justification that bed moisture is not a factor in classifying the high-rank coals which constitute overwhelmingly the major part of the world's total production of coal, and that the inclusion of bed moisture with such coals serves to invalidate to a degree the accuracy obtainable in the laboratory determination of one of the indexes (B.t.u.) plotted in Figs. 1 and 2, because bed moisture may vary at least 1 to 2 per cent in samples collected in the same mine. Such objections, if recognized as valid, may be taken care of by the use of the moisture-and-ash-free basis for high-rank coals and the as-received, ash-free basis for the low-rank coals, or for presenting the general relationships for the entire range of coals from lignite to anthracite.

That the differences in the geologic ages of various coals may or may not be regarded as sufficient cause for the variations exhibited in the physical and chemical properties of these coals is of interest in studying the numerous charts that appear in this paper. Of the coals plotted, those of Washington are of Tertiary (Eocene) age, while the older coals of the eastern states are of Paleozoic (Carboniferous) age; yet in physical and chemical properties, as measured by the criteria plotted in Figs. 1 and 2, the coals of Washington are much the same as older coals of equal rank. It is noted also that coals of the same geologic age and even coals of the same bed may vary widely in physical and chemical properties. Some investigators have pointed out the variations in the characteristic properties between young coals and older coals. This use of geologic age as a criterion of maturity may be satisfactory in general, but where natural, regional carbonization or metamorphism has taken place, rank rather than geologic age measures best the composition and characteristic properties of a coal.

FRIABILITY OF COAL

Until quite recently, the conception of the relative friabilities of different ranks of coals has consisted principally of abstract generalizations formed from experience in the handling of the coals. Several investigators have devised laboratory tests to measure the relative friability of coal by subjecting weighed samples of sized material to some external mechanical force and measuring the degradation in size, the amount of fines produced, etc. Stopes and Wheeler,[8] in England, ground sized coal in a ball mill and measured only the amount of dust formed which passed through a 200-mesh sieve (0.063-mm. opening). Smith,[9] working at the Engineering Experiment Station of the University of Illinois, determined friability by dropping 60 pieces of coal of 3 by 2½-in. round-hole size from a height of 10 ft. to a concrete floor and measuring the degradation by screening the resultant material through round-hole screens of 2½, 2, 1½ and ½-in. size. The degradation both in size and weight was computed and each expressed by a single number. The latter method is similar to the standard shatter test for coke specified by the American Society for Testing Materials.[10] More recently, Stansfield and associates[11] utilized the standard paving-brick testing rattler of the American Society for Testing Materials for testing the friability of coal. The method consisted in tumbling 2500 g. of 3-in. lumps of fresh coal for 3 min. in the standard rattler machine. The degradation in size of the resultant material is measured by screening on 2-in., 1-in., ½-in. and ¼-in. standard Tyler sieves. From these values an arbitrary strength index, the converse of a friability index, is calculated.

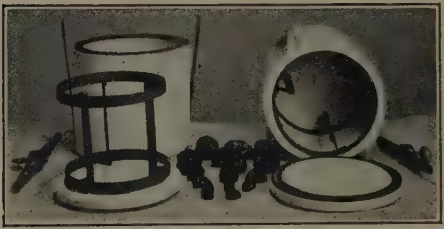

FIG. 3.—FRIABILITY JARS, FRAMES AND 1.5 TO 0.742-IN. SIZES OF COAL FROM A FRIABILITY DETERMINATION.

The method adopted by the authors is that of the Fuel Testing and Research Laboratories of the Department of Mines, Mines Branch, Canada.[12] The method and the results of the friability tests made on the coals included in this investigation are described in detail in an earlier Bureau of Mines report.[13] Briefly, the method is as follows: 1000 g. of fresh coal, sized to pass through a 1½-in. opening square-hole sieve and remain on a 1.05-in. opening square-hole sieve, is placed in a straight-sided porcelain jar mill approximately 7¼ in. deep and 7¼ in. dia., and the jar is rotated for 3 hr. at 40 r.p.m. (7200 turns as measured with a revolution counter). Fig. 3 shows the jar and the steel frame which supports three lifters for picking up and dropping the coal during rotation.

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[8] M. C. Stopes and R. V. Wheeler: The Spontaneous Combustion of Coal in Relation to Its Composition and Structure. *Fuels in Sci. and Prac.* (1923) **2**, 36.

[9] C. M. Smith: An Investigation of the Friability of Different Coals. Univ. of Ill. Eng. Exp. Sta. *Bull.* 196 (1929).

The Friability of Illinois Coals. Univ. of Ill. Eng. Exp. Sta. *Bull.* 218 (1930).

[10] American Society for Testing Materials (1927) Pt. II, 570.

[11] E. Stansfield, W. A. Lang, K. C. Gilbart, R. G. Brewer and M. F. Teskey: Eleventh Annual Report of the Research Council of Alberta, Fuels Division, *Rept.* 26 (1930) 18–20.

[12] J. H. H. Nicolls: Friability Tests of Various Fuels Sold in Canada. Canada Dept. of Mines, Mines Branch, Inv. of Fuels and Fuel Testing, *Bull.* 644 (1926) 20–35

The contents of the jar are sieved carefully by hand on the following Tyler square-mesh sieves: 1.05, 0.742, 0.525 and 0.371 in., and 4, 8, 14, 28, 48, 100 and 200 mesh. The weights of the screened products are then noted. From these data the average size is computed and a single friability value expressing percentage degradation in size is calculated as follows:

$$\text{Friability, per cent} = 100 \times \frac{D - d}{D}, \text{ in which}$$

D = average size of pieces before test,
d = average size of pieces after test.

The method of expressing friability as percentage degradation in size by a single figure was developed during the present investigation.

FRIABILITY AND RANK

Table 1 shows the average of the individual friability determinations on all the coals of each rank examined in comparison with the average ranking indexes—bed moisture content, fixed carbon content and heating value. There is a gradual increase in friability with increase in rank from lignite to semibituminous coal. The individual results for each coal are given in Figs. 4 to 10, inclusive, which show graphically the relation of friability, a physical property, and several indexes of rank. Friability, as expressed by the percentage degradation in size of the

TABLE 1.—*Average Friability Indexes of Different Ranks of Coal*

Property	Basis	Anthracite	Semibituminous	Bituminous	Subbituminous A	Subbituminous B	Lignite
Moisture, per cent	As received	5.2	2.6	4.4	13.3	26.0	37.3
Fixed carbon, per cent	As received, ash-free	90.8	75.8	55.7	46.6	35.8	32.6
Heating value, B.t.u.	As received, ash-free	13,990	15,210	14,050	11,470	9,020	7,370
Friability, per cent	As received	33.2	70.4	43.3	30.2	20.3	12.1
Number of individual tests		36	27	87	40	29	16

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[13] Reference of footnote 1.

different coals tested, is plotted on rectilinear graphs against each of the following: moisture on the as-received or bed-moisture basis (Fig. 4); British thermal units (Figs. 5 and 6) and fixed carbon (Figs. 7 and 8) on both the as-received, ash-free basis and the moisture-and-ash-free basis; and total carbon on both the as-received, ash-free, sulfur-free and nitrogen-free basis (Fig. 9), and on the moisture, ash, sulfur and nitrogen-free basis (Fig. 10).

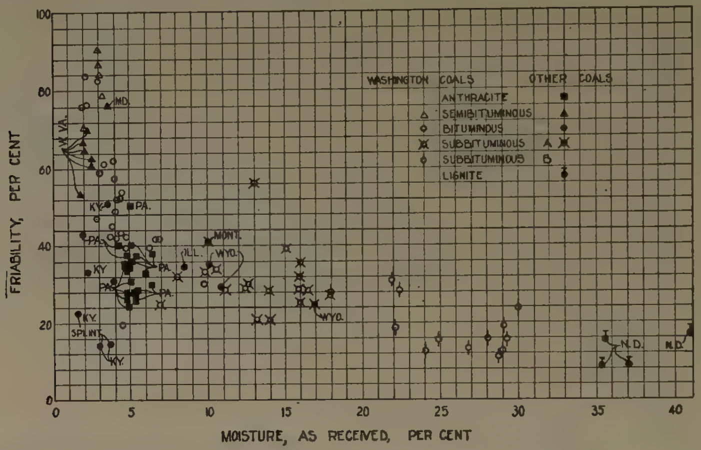

FIG. 4.—RELATION OF FRIABILITY AND MOISTURE CONTENT OF COAL, AS-RECEIVED BASIS.

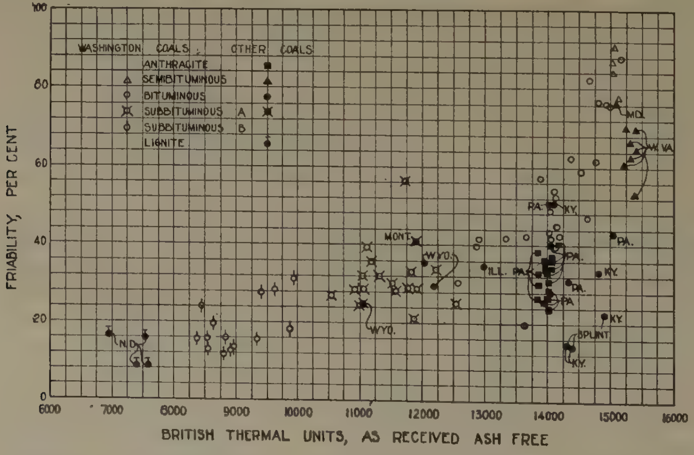

FIG. 5.—RELATION OF FRIABILITY AND BRITISH THERMAL UNITS, AS-RECEIVED, ASH-FREE BASIS.

It is at once evident from these graphs that there is a steady increase in friability from a minimum for North Dakota lignites through the subbituminous and bituminous coals to a maximum for Washington semibituminous coals, and then a rapid decrease through the West Virginia semibituminous coals to the Pennsylvania anthracites. This relationship is brought out more clearly in Fig. 7 (which shows fixed carbon content, as-received, ash-free basis, plotted against friability) than in any of the other figures. Here again, as in the classification charts (Figs. 1 and 2), it is evident that the use of the as-received, ash-free basis gives a wider distribution of the coals and less overlapping between different ranks than the use of the moisture-and-ash-free basis. In the charts showing friability plotted against bed moisture (Fig. 4) and heating value (Figs. 5 and 6), the points representing the anthracites, which

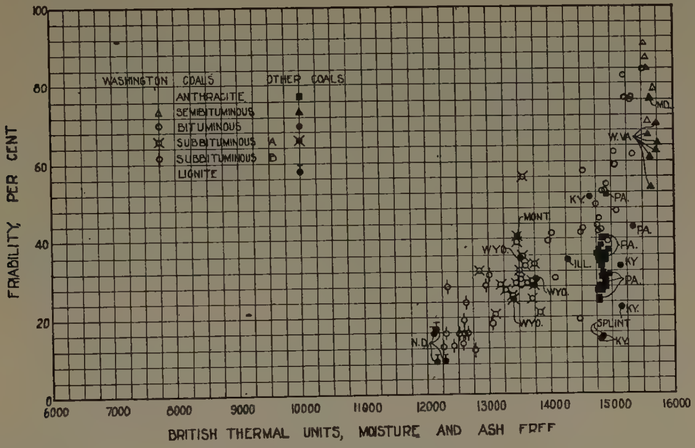

FIG. 6.—RELATION OF FRIABILITY AND BRITISH THERMAL UNITS, MOISTURE-AND-ASH-FREE BASIS.

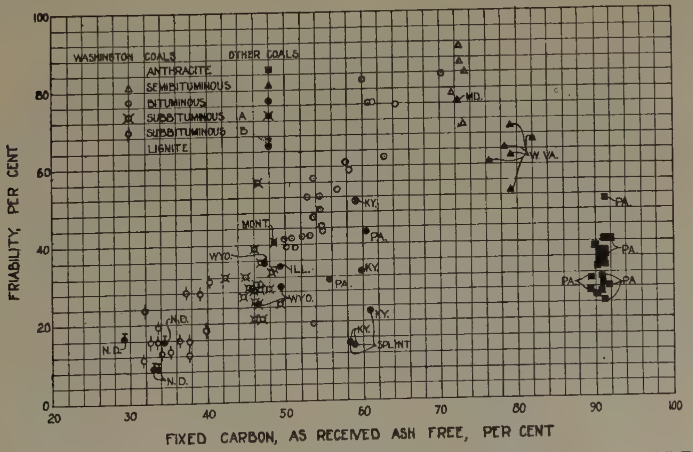

FIG. 7.—RELATION OF FRIABILITY AND FIXED CARBON CONTENT OF COAL, AS-RECEIVED, ASH-FREE BASIS.

have an average friability of 33.2 per cent, naturally lie in the area occupied by the less friable bituminous coals and hence are not sharply differentiated from them, but when friability is plotted against fixed carbon or against total carbon of the ultimate analysis (Fig. 9), especially on the as-received, ash-free basis, the differentiation is marked. It is more marked with fixed carbon than with total carbon because the range in fixed carbon content of the coals is greater than the range in total carbon content.

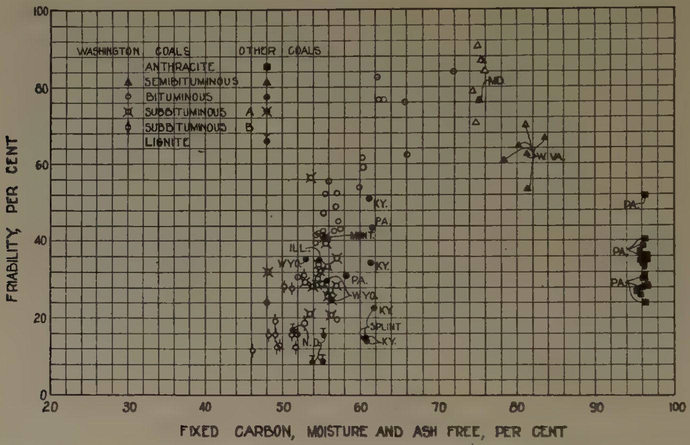

FIG. 8.—RELATION OF FRIABILITY AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

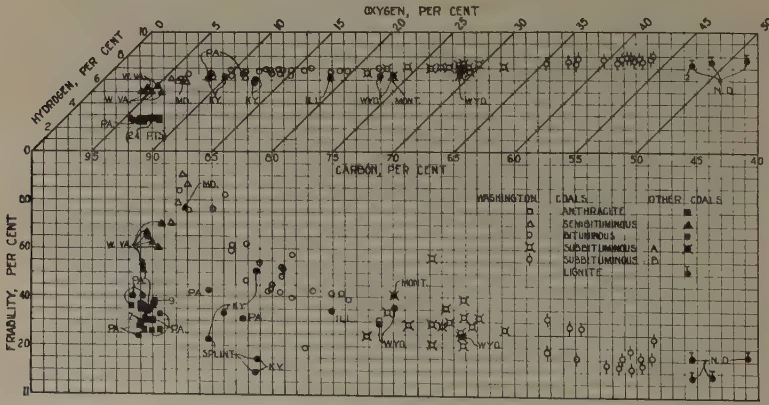

FIG. 9.—RELATION OF FRIABILITY AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

The anthracites from the eastern middle field in Pennsylvania examined in this study gave somewhat anomalous results with respect to friability; that is, they deviated from expectations in showing higher average friabilities than the subbituminous coals and the lignites. The friabilities determined on the anthracites ranged from 24.0 to 51.6 per cent, with an average for all samples of 33.2 per cent. Anthracite is considered to be the hardest of all coals, and if this be true, it is apparent that the method of estimating friability used in this investigation is not a measure of hardness alone. It seems evident that the anthracites gave higher friability values than the subbituminous coals and lignites because they are more brittle. The subbituminous coals and lignites, on the other hand, although less brittle than the anthracites, exhibit greater elasticity and cohesion.

It seems possible that samples of anthracite from the less disturbed areas in the northern field of Pennsylvania may show lower friabilities than those from the eastern middle field reported herein. Samples from all of the anthracite fields will be collected and the results reported in a subsequent publication.

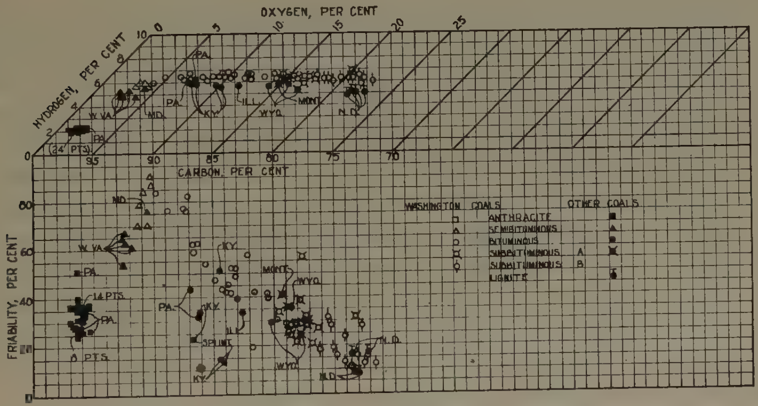

Fig. 10.—Relation of friability and ultimate analyses, moisture, ash, sulfur and nitrogen-free basis.

It is obvious that friability is not a major factor in determining rank for purposes of classification, although generally there is a relation between friability and rank. The range in friability for each rank is too great, and specific coals of both the lower and higher ranks have similar friabilities. Consequently, coals having friabilities between 25 and 30 may be of either subbituminous, bituminous or anthracite rank.

The friability of a given coal appears to depend to a large extent upon the nature of the metamorphic forces to which it has been subjected. In areas where tilting, folding and faulting of the beds has occurred, the friability of the coal is generally increased. Where such action has been recent and local, as in Washington, the resulting high-rank coals are generally extremely friable.

SLACKING CHARACTERISTICS OF COAL

The characteristic criterion, according to the U. S. Geological Survey, by which bituminous and subbituminous coals are differentiated is their behavior upon weathering. Drying alone or alternate wetting and drying causes subbituminous coals to disintegrate or "slack," but bituminous coals do not exhibit this characteristic. In order to compare the slacking characteristics of different coals, an accelerated laboratory slacking test was developed at the Pittsburgh Experiment Station of the Bureau of Mines by Fieldner, Selvig and Frederic. The Washington coals and a number from other states were tested at the Pittsburgh Station by this method. The detailed results of this work appear in a recent Bureau of Mines report,[14] from which the major part of the following discussion of slacking characteristics is taken.

The accelerated slacking test is described briefly as follows: About 500 to 600 g. of fresh coal, sized to pass through a 1½-in. opening square-hole sieve and remain on a 1.05-in. opening square-hole sieve, are shaken according to a regular procedure on an 8-in. sieve having 0.263-in. square openings. The percentage of undersize is designated as the "blank sieving." The oversize is dried at 30° to 35° C. for 24 hr., allowed to cool to room temperature and then immersed in water at room temperature for 1 hr. After the water has been carefully drained off the sample is again dried at 30° to 35° C. for 24 hr., cooled and sieved in the same manner as prescribed for the blank sieving. The percentage of undersize less the blank sieving is designated as the first cycle slacking index. The procedure of wetting, drying and sieving is continued for six cycles, or until all the sample has passed through the sieve.

SLACKING CHARACTERISTICS AND RANK

Figs. 11 to 15 inclusive show graphically the relationship between the slacking characteristics of the coals examined and several indexes of rank. First cycle slacking indexes are plotted against the following: moisture on the as-received basis (Fig. 11); fixed carbon both on the as-received, ash-free basis (Fig. 12) and on the moisture-and-ash-free basis (Fig. 13); total carbon both on the as-received, ash-free, sulfur-free and nitrogen-free basis (Fig. 14) and on the moisture, ash, sulfur and nitrogen-free basis (Fig. 15).

Fig. 11 shows that there is a general relationship between the slacking characteristics of coal and rank as indicated by bed moisture content. The general tendency shown in this chart is for the slacking indexes to increase with increasing bed moisture content, but, as is evident, specific coals of the same moisture content may exhibit widely differing slacking characteristics. Hence, bed moisture content is not necessarily a reliable index of the slacking characteristics of a given coal, as has been suggested by other investigators, and should not be used as a substitute for the slacking test itself. The averages of the slacking indexes shown in Fig. 11 for coals of semibituminous, bituminous, subbituminous A, subbituminous B, and lignite ranks are 1.0, 2.8, 10.3, 48.5 and 65.0, respectively.

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[14] A. C. Fieldner, W. A. Selvig and W. H. Frederic: Accelerated Laboratory Test for Determination of Slacking Characteristics of Coal. U. S. Bur. Mines *Rept. of Investigation* 3055 (1930).

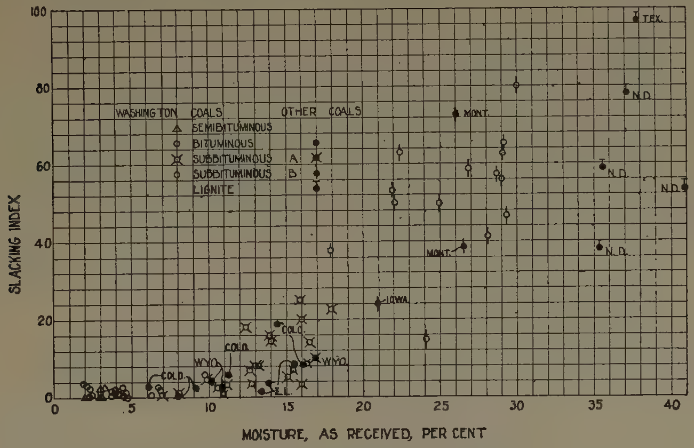

FIG. 11.—RELATION OF FIRST-CYCLE SLACKING INDEXES AND MOISTURE CONTENT OF COAL, AS-RECEIVED BASIS.

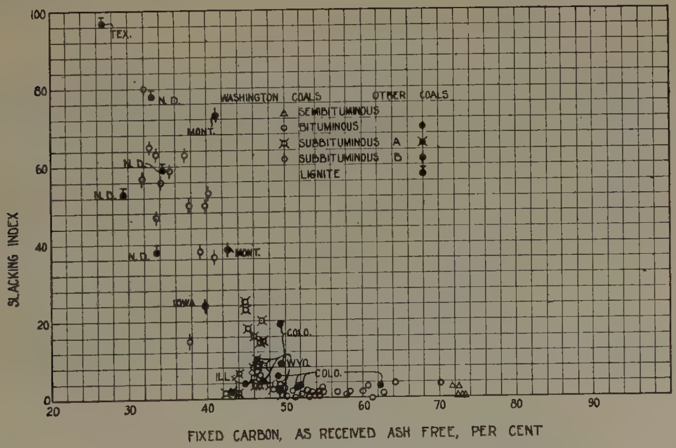

FIG. 12.—RELATION OF FIRST-CYCLE SLACKING INDEXES AND FIXED CARBON CONTENT OF COAL, AS-RECEIVED, ASH-FREE BASIS.

Figs. 12 and 13 show the relationship between slacking index and fixed carbon; and Figs. 14 and 15 show similar relations with total carbon. In Figs. 12 and 14, the fixed and total carbon, respectively, are plotted on the as-received basis, and in Figs. 13 and 15, the same indexes of rank are plotted on the moisture-free basis. In general, there is a better disposition of points when the bed moisture is included, as, for example, in Fig.

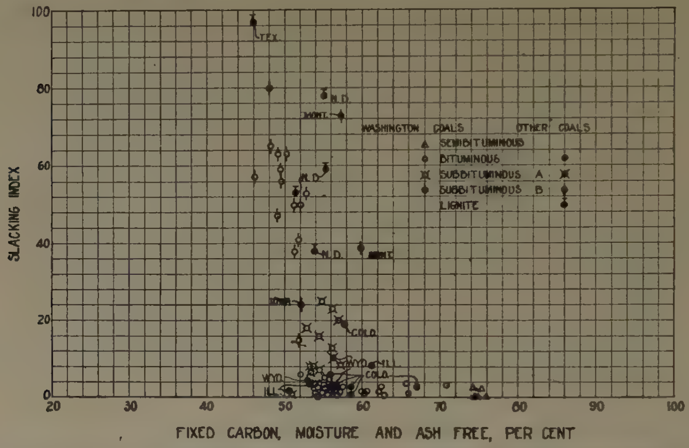

FIG. 13.—RELATION OF FIRST-CYCLE SLACKING INDEXES AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

12, all the coals having more than 50 per cent fixed carbon content have slacking indexes of less than 10; all those with fixed carbon contents between 44 and 50 per cent have slacking indexes of less than 28; and all the coals, except four, having less than 44 per cent fixed carbon content have slacking indexes above 28. On the other hand, in Fig. 13, where bed moisture is excluded, more than one-half of the coals plotted fall

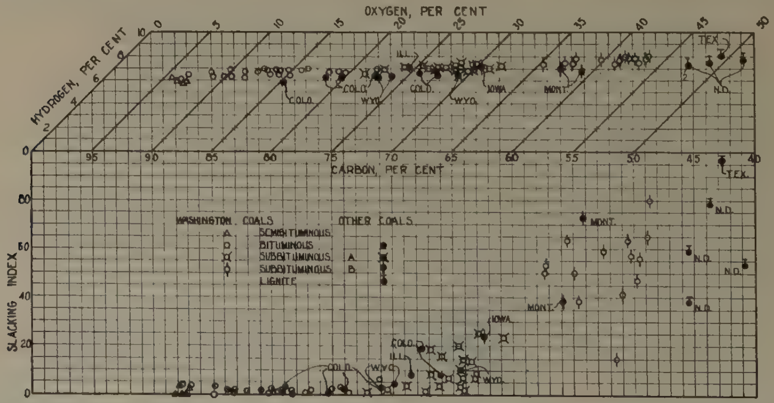

FIG. 14.—RELATION OF FIRST-CYCLE SLACKING INDEXES AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

between 52 and 58 per cent fixed carbon content, vary in rank from bituminous to lignite, and vary in slacking index from 0 to nearly 80. The exclusion of bed moisture has a similar but less pronounced distorting effect on the arrangement of points when slacking index is plotted against total carbon (Fig. 15).

As already pointed out, the principal characteristic used to differentiate between subbituminous and bituminous coal according to the U. S. Geological Survey classification is that of slacking. Subbituminous coals exhibit slacking characteristics, but bituminous coals do not. In general, this distinction is borne out by the data on slacking presented here, but, as in all other attempts to organize and classify the properties of this natural metamorphic product, there are several coals that exhibit reactionary tendencies. For example, the coals from the Bellingham and New Black Diamond mines in Washington have long been considered as a subbituminous rank, yet both gave low first-cycle slacking indexes characteristic of bituminous coals; the indexes were 0.9 and 3.0 respectively. According to heating value and fixed carbon content both coals would normally be of subbituminous rank. It might be considered that the results of the slacking test at the sixth cycle of treatment would show whether these coals exhibit the characteristics typical of subbituminous coal, but this is not so. At the end of the third and the sixth cycle, the Bellingham and New Black Diamond coals showed slacking indexes of 3.6 and 9.1, and 10 and 20 respectively. Values as high as these are obtained for very friable bituminous coking coals, due largely to the difficulty of correcting intelligently the slacking indexes for friability at the end of successive cycles of treatment.

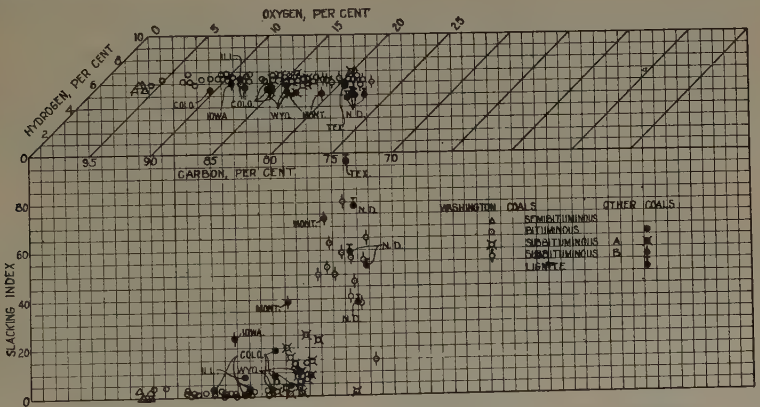

FIG. 15.—RELATION OF FIRST-CYCLE SLACKING INDEXES AND ULTIMATE ANALYSES, MOISTURE, ASH, SULFUR AND NITROGEN-FREE BASIS.

LOW-TEMPERATURE ASSAY OF COAL

Low-temperature assays were made by the Fischer method[15] on 63 coals from Washington and 19 coals from other states. A description of the test with the results of the assays is given in detail in a Bureau of Mines report.[16] In a previous investigation, Davis and Galloway[17] compared the results of low-temperature assays of 19 subbituminous coals and lignites by three different methods, and found that the Fischer method gave the most reliable results. A brief description of the low-temperature assay method is as follows: 250 g. of air-dried coal, ground to pass a 10-mesh sieve, is heated in a Fischer cylindrical, aluminum retort until a maximum temperature of 550° C. has been reached, usually in about 75 minutes. The temperature is maintained at 550° C. until the evolution of gas has become very slow, which occurs usually after 2¾ to 3¼ hr. of heating. The water, gas, tar and oil are collected in appropriate receivers and the amount of each product is determined. The gas is analyzed by an Orsat gas apparatus.

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[15] F. Fischer and H. Schrader: Crude Tar Determination with an Aluminum Distilling Apparatus. *Ztsch. f. angew. Chem.* (1920) **33**, 172–175.

Low-temperature Assay Yields and Rank

Figs. 16 to 27 show graphically the relation between rank and low-temperature assay yields of coke, gas, tar and oil. The percentage yields of each of these low-temperature products are plotted against the following: fixed carbon on the as-received, ash-free basis and on the moisture-and-ash-free basis; total carbon both on the as-received, ash-free, sulfur-free and nitrogen-free basis and on the moisture, ash, sulfur and nitrogen-free basis.

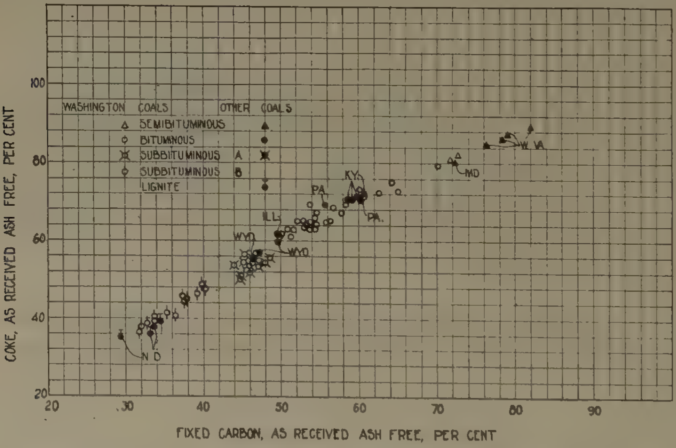

Fig. 16.—Relation of yield of low-temperature coke and fixed carbon content of coal, as-received, ash-free basis.

Of the low-temperature products, the coke yield on an as-received, ash-free basis as plotted in Figs. 16 and 18 against fixed and total carbon, respectively, shows the best relationship with rank (as expressed by fixed carbon and total carbon content), although in Fig. 16 there is some

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[16] Reference of footnote 1.

[17] J. D. Davis and A. E. Galloway: Low-Temperature Carbonization of Lignites and Subbituminous Coals. *Ind. & Eng. Chem.* (1928) **20**, 612.

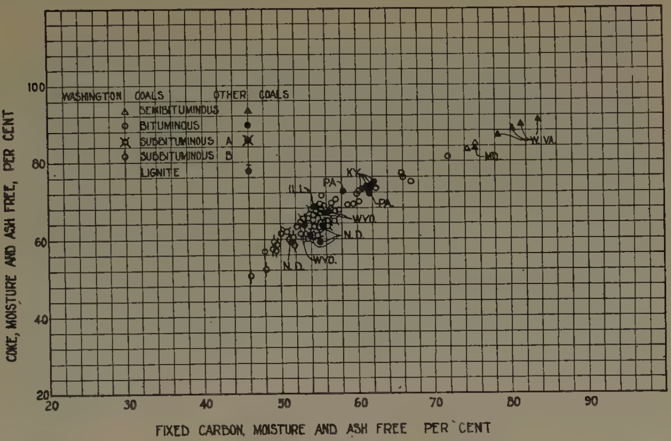

FIG. 17.—RELATION OF YIELD OF LOW-TEMPERATURE COKE AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

overlapping between the lignites and subbituminous B and between the subbituminous A and bituminous ranks. Fig. 18 shows a similar overlapping between the subbituminous A and bituminous ranks and between the bituminous and semibituminous ranks.

A smoothed curve which approaches closely a straight line may be drawn through the points in Fig. 16; the relationship is exactly linear for

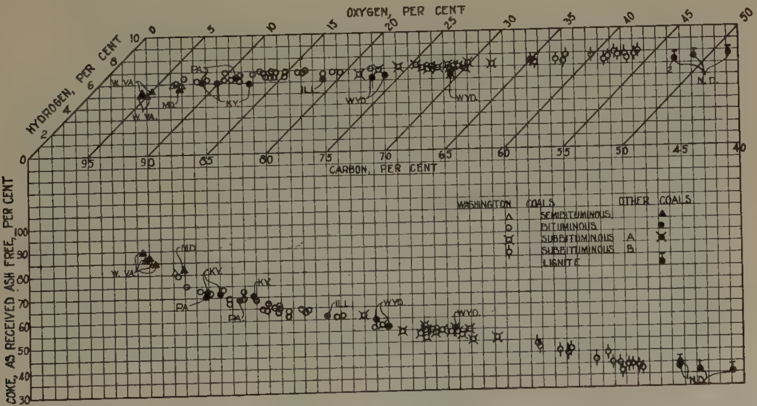

FIG. 18.—RELATION OF YIELD OF LOW-TEMPERATURE COKE, AS-RECEIVED, ASH-FREE BASIS, AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

the points representing coals from bituminous to lignite, inclusive. A general relationship between the yield of low-temperature coke and the fixed carbon content of the coals would be expected, of course, but the agreement shown in Fig. 16 for the whole range is surprisingly close if one considers that the heating rate and maximum temperature in the volatile-matter determination are quite different from those in the low-temperature assay. Fig. 16 shows that all the semibituminous coals examined yielded more than 80 per cent of low-temperature coke and that all but one of the bituminous coals yielded between 56 and 80 per cent of low-temperature coke.

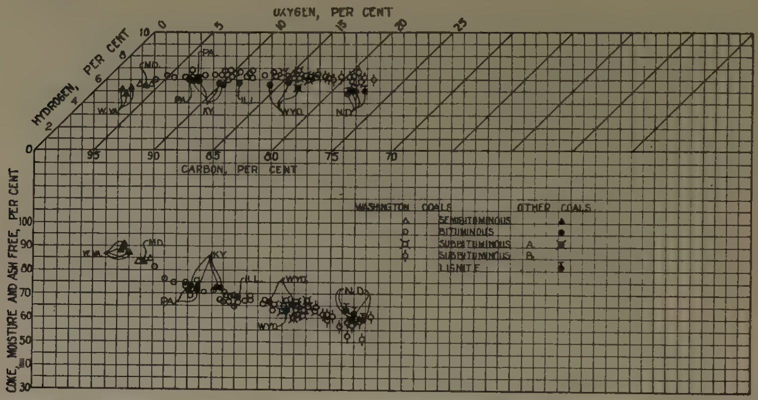

FIG. 19.—RELATION OF YIELD OF LOW-TEMPERATURE COKE, MOISTURE-AND-ASH-FREE BASIS, AND ULTIMATE ANALYSES, MOISTURE, ASH, SULFUR AND NITROGEN-FREE BASIS.

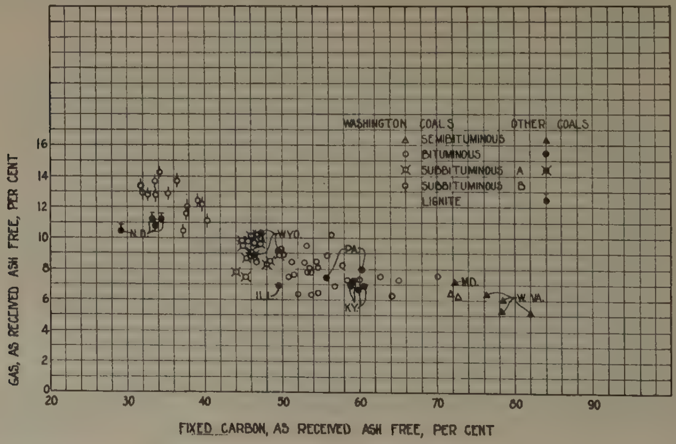

FIG. 20.—RELATION OF YIELD OF LOW-TEMPERATURE GAS AND FIXED CARBON CONTENT OF COAL, AS-RECEIVED, ASH-FREE BASIS.

In Figs. 17 and 19, in which bed moisture is excluded, the relative positions of the semibituminous and bituminous coals have not changed to any great extent because they are low-moisture coals, but those of all the lower ranks have changed to such an extent that there is almost no separation between the different lower ranks; several of the subbituminous A coals have moved up into the bituminous range.

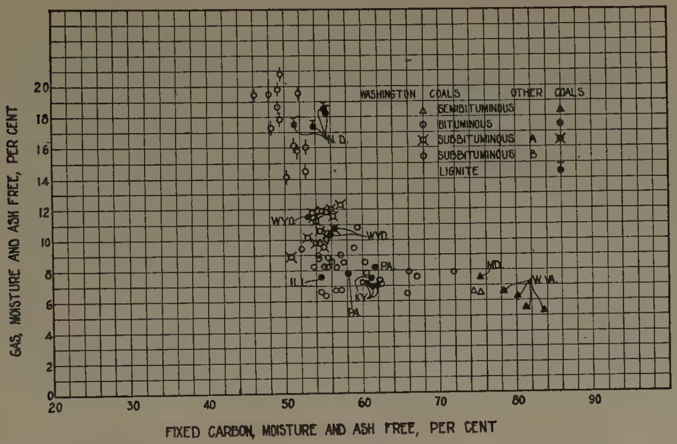

FIG. 21.—RELATION OF YIELD OF LOW-TEMPERATURE GAS AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

The yields of low-temperature gas are plotted against fixed and total carbon in Figs. 20 to 23, inclusive, which show graphically the relation between the yield of low-temperature gas and rank. In general, the yields of low-temperature gas increase with decreasing rank to a maximum

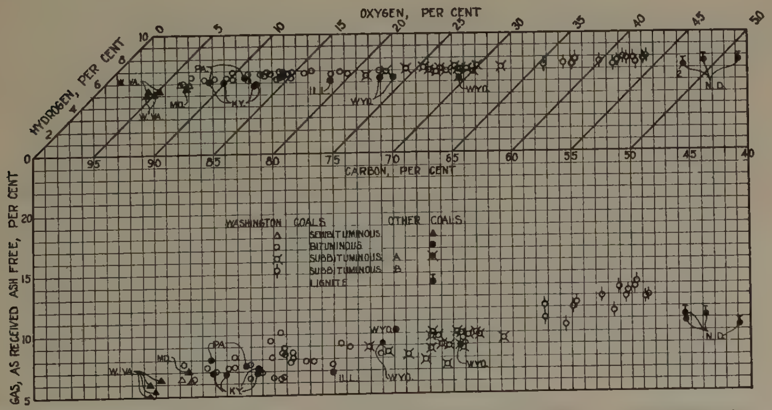

FIG. 22.—RELATION OF YIELD OF LOW-TEMPERATURE GAS, AS-RECEIVED, ASH-FREE BASIS, AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

for the subbituminous B coals and then decrease slightly to the lignites. As is well known, coals of the same rank show considerable variation in gas-forming properties. The graphs show that coals of several different ranks have similar low-temperature gas-forming properties with respect to their weight yields, but the composition of the gas, of course, varies widely for the different ranks. The following changes in the composition by volume of low-temperature gas with the change from the lower to the higher rank coals were noted: (1) The percentage of carbon dioxide decreased with almost a corresponding but opposite change in the calorific value; (2) the percentage of saturated hydrocarbons increased; (3) there is a general tendency for the hydrogen to increase and for the carbon monoxide to decrease as rank increases; (4) there is little change in the percentage of unsaturated hydrocarbons with change in rank.

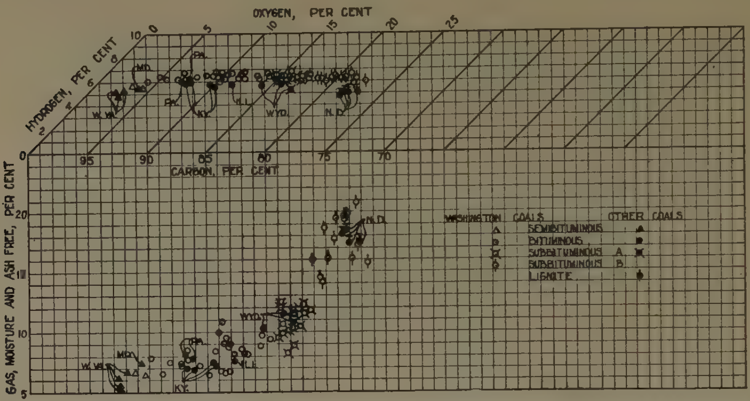

Fig. 23.—Relation of yield of low-temperature gas, moisture-and-ash-free basis, and ultimate analyses, moisture, ash, sulfur and nitrogen-free basis.

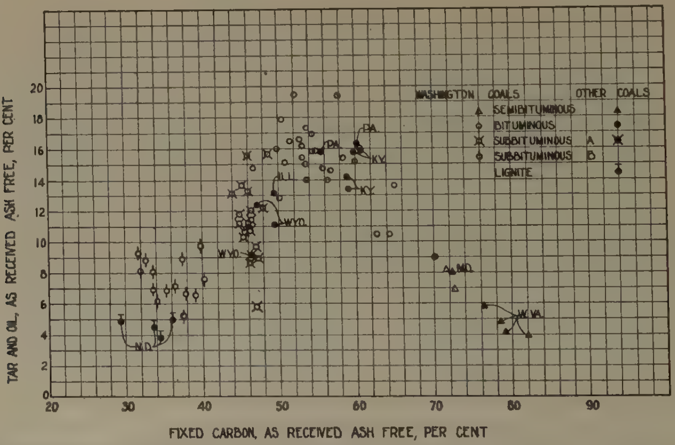

Fig. 24.—Relation of yield of low-temperature tar and oil and fixed carbon content of coal, as-received, ash-free basis.

The relationship between rank and the yields of low-temperature tar and oil are shown graphically in Figs. 24 to 27, inclusive, in a manner similar to that for the coke and gas yields. On an as-received, ash-free basis, as plotted in Figs. 24 and 26, the lignites, the lower rank subbituminous coals and the semibituminous coals give the lowest and about equal quantities of tar and oil. The yields reach a maximum with the bituminous coals having the composition corresponding to that of good gas coals. On a moisture-and-ash-free basis, as plotted in Figs. 25 and 27, the same general relationship is shown, but the yields of the low-rank coals are generally higher by several per cent, owing to the exclusion of

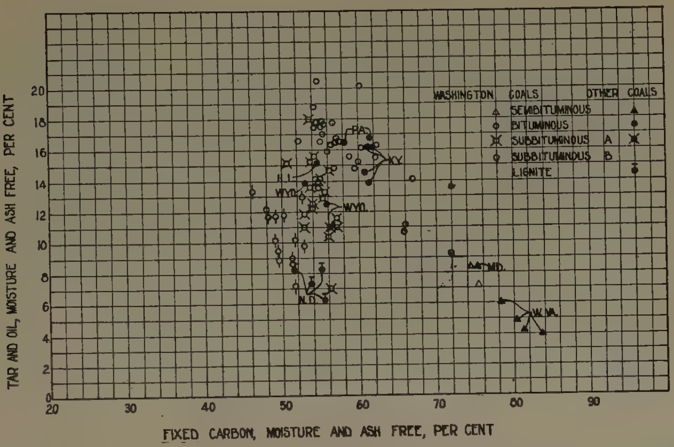

FIG. 25.—RELATION OF YIELD OF LOW-TEMPERATURE TAR AND OIL AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

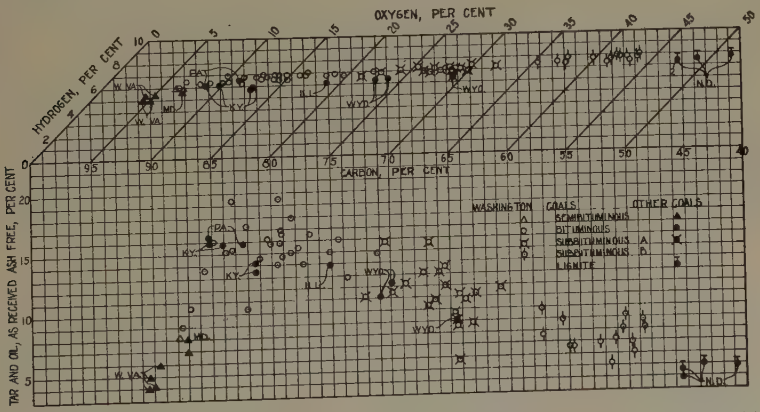

FIG. 26.—RELATION OF YIELD OF LOW-TEMPERATURE TAR AND OIL, AS-RECEIVED, ASH-FREE BASIS, AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

moisture, while those of the high-rank, low-moisture coals, especially the semibituminous and bituminous coals, have very nearly the same yields as on the as-received, ash-free basis.

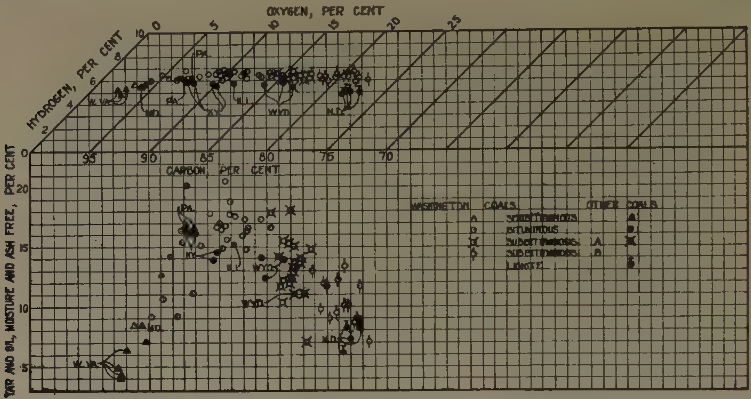

FIG. 27.—RELATION OF YIELD OF LOW-TEMPERATURE TAR AND OIL, MOISTURE-AND-ASH-FREE BASIS, AND ULTIMATE ANALYSES, MOISTURE, ASH, SULFUR AND NITROGEN-FREE BASIS.

AGGLUTINATING VALUE OF COAL

Of the coals studied in this investigation, 36 of bituminous and semibituminous rank gave appreciable agglutinating values as determined by an empirical laboratory method perfected by Marshall and Bird[18] at the Northwest Experiment Station of the United States Bureau of Mines. The method used is, briefly, as follows: An intimate mixture of 2.275 g. of air-dried coal, sized to pass a 100-mesh sieve (0.147 mm.), 22.725 g. of Ottawa sand, and one drop of glycerin is made, compressed in a crucible, and carbonized at 950° C. for 20 min. The sand is sized between 40 and 50-mesh sieves with openings of 0.381 and 0.279 mm., respectively. The resulting coke buttons are crushed by means of a Riehlé testing machine. The average compressive strength in grams of 10 buttons is considered as the agglutinating value.

AGGLUTINATING VALUE AND RANK

Figs. 28 to 31, inclusive, show the relationship between agglutinating value and rank. In general, the higher rank coals have the higher agglutinating values, but with these low-moisture coals the advantages, heretofore apparent in the charts plotted on as-received basis, disappear. The relationships on the as-received and moisture-free bases are equally satisfactory.

The semibituminous coals vary in agglutinating values from 7130 to 13,900 with an average of 11,110 for seven samples, and the bituminous coals from no appreciable agglutinating value to 9810 with an average of 3780 for all the samples which had appreciable agglutinating values. The agglutinating value of coal may be used in a limited way to determine rank, in that a coal with an agglutinating value of 10,000 and above is very likely to be a semibituminous coal and a coal having an intermediate or low agglutinating value to be of bituminous rank.

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[18] S. M. Marshall and B. M. Bird: Test for Measuring the Agglutinating Power of Coal. *Trans.* A. I. M. E., Coal Div. (1930) 340–383.

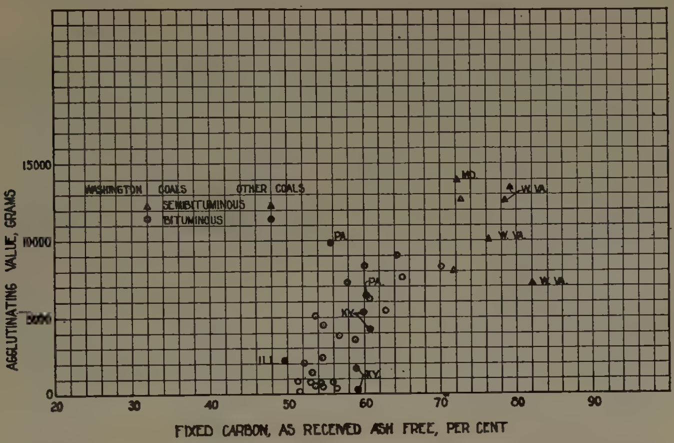

FIG. 28.—RELATION OF AGGLUTINATING VALUE AND FIXED CARBON CONTENT OF COAL, AS-RECEIVED, ASH-FREE BASIS.

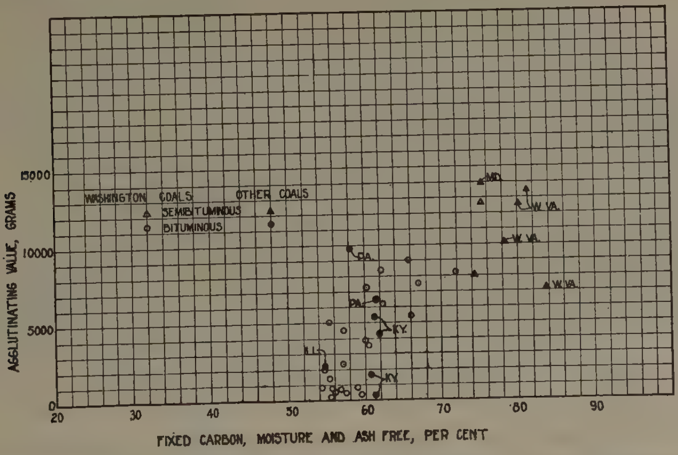

FIG. 29.—RELATION OF AGGLUTINATING VALUE AND FIXED CARBON CONTENT OF COAL, MOISTURE-AND-ASH-FREE BASIS.

The authors believe that the agglutinating test will prove of considerable value in classifying coals of questionable rank when it is applied to them. The agglutinating test was applied to all the Washington coals examined in this study, but, as would be expected, none below bituminous rank gave appreciable values. Some of the high-moisture coals of Illinois and Iowa have produced coke under certain carbonizing conditions. One of these, a so-called subbituminous coal from Taylor County, Iowa, when tested for slacking characteristics by Fieldner, Selvig and Frederic,

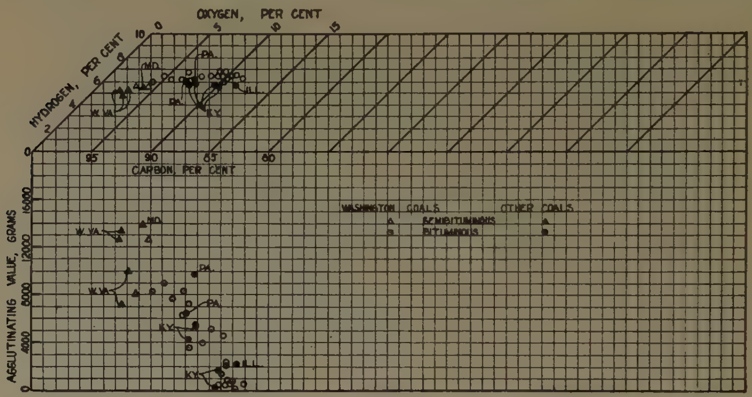

FIG. 30.—RELATION OF AGGLUTINATING VALUE AND ULTIMATE ANALYSES, AS-RECEIVED, ASH-FREE, SULFUR-FREE AND NITROGEN-FREE BASIS.

gave a first-cycle index of 24 per cent, yet a sample from the same mine gave a mid-temperature coke of fair quality when carbonized by the Parr process in an experimental retort at the University of Iowa.[19] Unfortunately, the agglutinating value of this coal was not determined in the present investigation. Presumably, the coal should show an appreciable

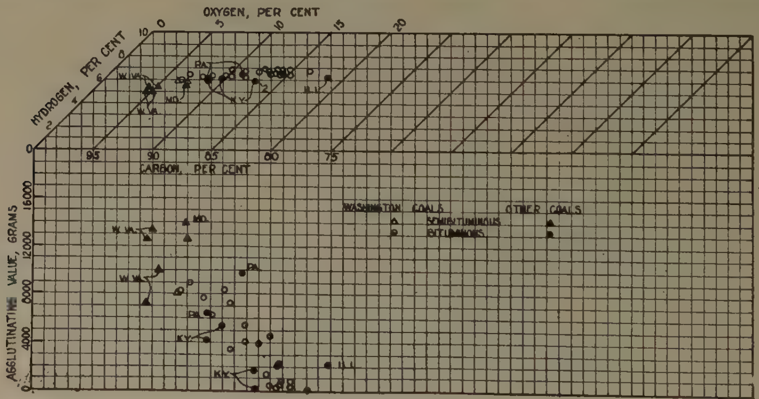

FIG. 31.—RELATION OF AGGLUTINATING VALUE AND ULTIMATE ANALYSES, MOISTURE ASH, SULFUR AND NITROGEN-FREE BASIS.

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[19] This Taylor County, Iowa, coal was found to have a "weathering coefficient" of 65 when tested by an accelerated method devised and reported by H. L. Olin, F. C. Johnson, Jr., R. C. Kinne, J. D. Waddell and J. N. Ambrose: Coking and Weathering Properties of Iowa Coals. Iowa Geol. Survey *Tech. Paper* 2 (1930) 30 and 52.

See also S. W. Parr and D. R. Mitchell: Slacking of Coal and Its Proper Interpretation. *Jnl. Ind. & Eng. Chem.* (1930) **22**, 1212.

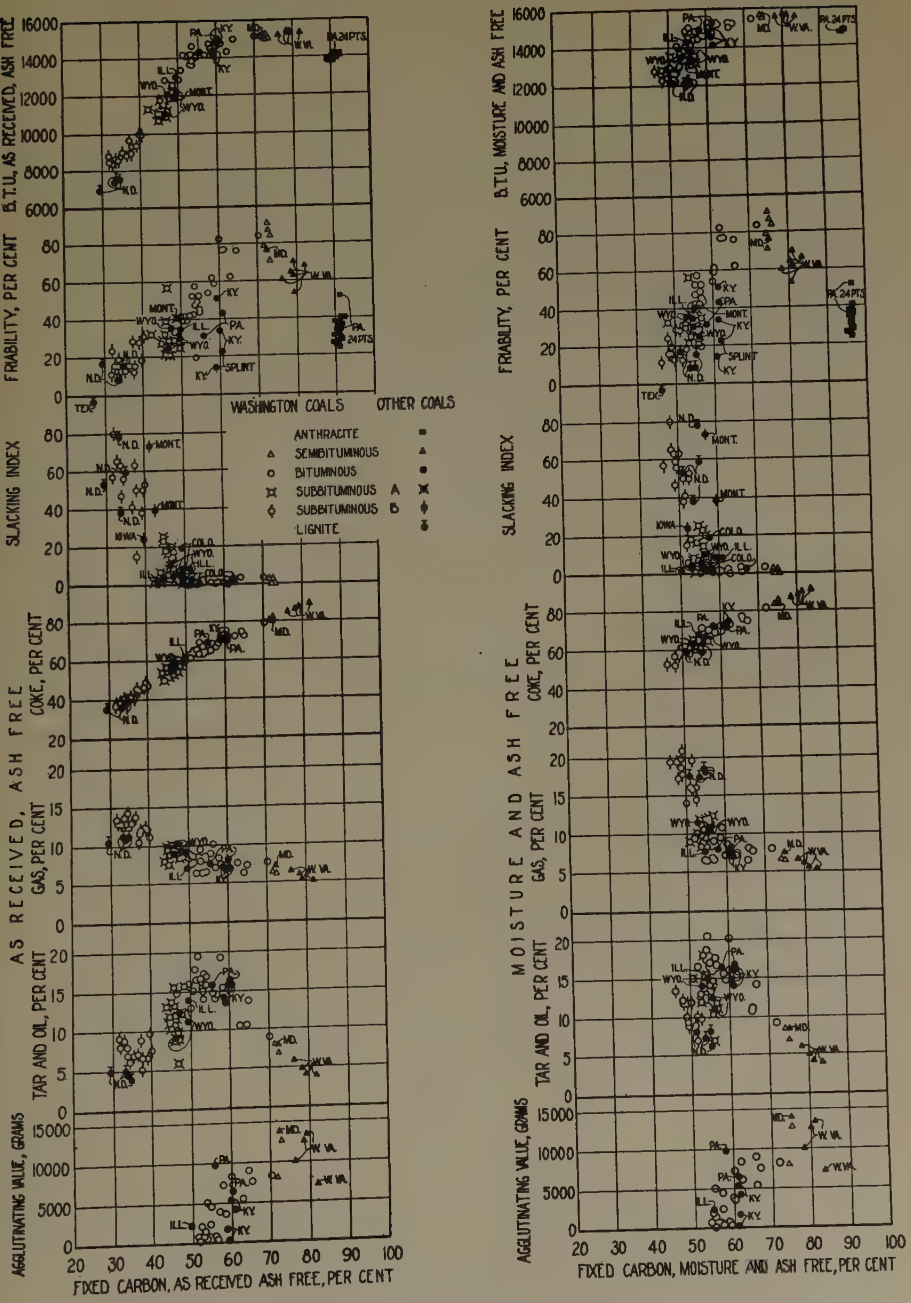

FIG. 32.—COMPARISON OF AS-RECEIVED AND MOISTURE-AND-ASH-FREE BASES FOR CLASSIFYING COALS.

value, since it produced a coke; hence it is believed that the agglutinating test should prove useful in classifying coals of doubtful rank.

Summary and Conclusions

This report presents a correlation in graphic form of two systems of coal classification with the results of a laboratory investigation of the friability, slacking characteristics, low-temperature carbonization and agglutinating property of approximately 100 coals from Washington and other states. One of these systems is based on the proximate analysis and the other on the ultimate analysis.

The various coals included in the investigation ranged in rank from lignite to anthracite. In the classification based upon the proximate analysis, fixed carbon content and heating value were used as ranking indexes; in the ultimate classification carbon, hydrogen and oxygen were used. In each case the results are shown on both the as-received, ash-free basis and the moisture-and-ash-free basis.

The classification based on fixed carbon content and heating value is preferred to that based on carbon, hydrogen and oxygen, because proximate analyses are more readily available and because the range in fixed carbon content from low to high-rank coals is greater than the range in total carbon content. With respect to the bases upon which the analyses are plotted (Fig. 32), the as-received—that is, bed moisture basis—is considered superior to the moisture-free basis, not only for presenting the relationships between the different ranks of coal from lignite to anthracite but also for correlating the relationships between rank and the physical and chemical properties studied in this investigation—friability, slacking characteristics, and yield of products on carbonization. The use of the as-received, ash-free basis for classification purposes (1) results in less overlapping among coals of questionable rank, (2) represents more nearly the commercial product as marketed, and (3) portrays more satisfactorily the successive changes to which the coal-forming material has been subjected and the physical and chemical properties of the whole range of products resulting from these changes than does a classification based upon coal calculated free of both moisture and ash.

Acknowledgments

This investigation was suggested by and carried out under the general direction of A. C. Fieldner, chief engineer, Experiment Stations Division, U. S. Bureau of Mines, to whom the authors are indebted for numerous suggestions given throughout the course of the work. The samples of West Virginia coals were collected through the courtesy of M. R. Campbell, senior geologist, U. S. Geological Survey.

DISCUSSION

(H. J. Rose presiding)

D. R. Mitchell and C. M. Smith, Urbana, Ill. (written discussion).—The attempt of Messrs. Yancey and Johnson to correlate physical and chemical properties of coal in relation to classification of coal into types has considerable merit. In general their results agree with similar tests that have been made at the University of Illinois, mainly on Illinois coal.

Irregularities in chemical and physical results are many, although the irregularities are much more marked for the physical properties. It is sometimes extremely difficult to explain these variations in physical properties, which may be found in samples of coal from a single mine showing consistent chemical values. In regard to slacking, only broad generalizations can be made. It is probable that many of the variations noted are due not so much to actual changes in the coal substance as to such other factors as irregular sampling technique, lack of knowledge of the condition of the coal when sampled, presence of impurities and faulty testing procedure.

At one mine in northern Illinois three samples were taken for slacking tests in different sections of the mine. The bed moisture of the samples was 12.0, 10.9 and 15.7, with corresponding slacking indices after the eighth cycle of 21.9, 23.4 and 15.3. These results aroused our interest, particularly because the normal coal-bed moisture content in this district was from 15 to 16 per cent. Further investigation revealed that the sections of the mine where the first two samples were taken had not been worked for some years until a few days before the samples were taken. Evidently there was some loss of moisture at some distance back from the working face, which not only lowered the normal coal-bed moisture content but had some effect on the slacking characteristics of these samples.

As to friability, one gets the impression from Fig. 4 that moisture is the controlling factor in the friability of the subbituminous coals and lignite, but that it has little or nothing to do with the friability of the higher rank coals. This confirms our experience with respect to Illinois bituminous coals, as we were unable to establish a correlation between friability and moisture in drop tests on more than 50 samples, from different seams and districts within the state.

It seems a little odd to find the lower ranking coals, subbituminous and lignite, less friable than most of the higher ranking ones, including anthracite. While we have never drop-tested the low-rank coals, we found Illinois bituminous coal much more friable than Pennsylvania anthracite, which showed a size degradation of about 12 per cent in our tests. This is comparable with the average friability of 33.2 per cent for anthracite coal, cited by the authors (p. 180). Their point about the greater elasticity and cohesion of the low-rank coals seems to be well taken. Perhaps the high moisture content is partly, if not largely, responsible for this. May it not be that the tumbling jar test fails to separate or break down the cohesive particles beyond a certain limit, and thereby fails to reveal the true friability of these coals?

This emphasizes the need for comparing the numerous methods which have been developed for testing the friability of coals by subjecting samples of a number of different coals to each kind of test. Such a study should lead to the adoption of a standard test, which could be relied upon to give a true indication of coal friability.

Attempts made to iron out some of these irregularities in slacking and friability results by using clean coal obtained by float and sink separation at a certain specific gravity have not been entirely satisfactory. Certainly, if it is assumed that pure coal of a given rank has certain definite chemical and physical properties, samples used for physical tests should be as near as possible to the pure coal substance as found in nature.

However, the authors' retention of moisture as a partial basis for classification in Figs. 7, 11, 14, etc., is to be commended, as this helps to distribute the coals over a wider range.

A list of the analyses of these coals would help the reader in tying up the physical and chemical properties of the coals with the many plotted diagrams.

R. S. Julsrud, Philadelphia, Pa. (written discussion).—The writer notes that the samples of anthracite selected for study were taken from the middle eastern field of the anthracite region in Pennsylvania. R. M. Hardgrove, in April, 1931, gave the results of a number of grindability tests on anthracite from the several Pennsylvania fields.[20] A bituminous coal from the Upper Kittanning seam in Somerset County, Pennsylvania, was selected as representing 100 grindability. It was found that the anthracites sampled showed a remarkably uniform variation from an average grindability of 21 in the eastern end of the northern field to 52 in the western southern field. It appears that the grindability of his samples bear no marked relationship to either the fixed carbon or the volatile content. It is suggested that possibly the specific gravity of coal bears some relation to its friability and grindability. It will be interesting to learn of Yancey and Johnson's further work on anthracites from the northern and southern fields as a means of comparing the grindability and friability of these fuels.

G. H. Cady, Urbana, Ill. (written discussion).—The authors note a tendency towards friability to increase from rank to rank. Is this generalization based upon a sufficient variety of coals to justify its general acceptance, or is it a generalization applicable particularly to coals from Washington in which coals vary rapidly in rank, due primarily to tangential pressures? In such a region jointing might be developed progressively with the age of the coal, whereas in regions of flat-lying undisturbed rocks where vertical pressure was the main consolidating influence jointing might not be so conspicuously developed at the same stages. I believe that in Illinois coals friability is somewhat greater for the lower than for the higher rank coals. Smith states[21] that weight degradation of No. 6 coal (all in southern Illinois) is 37.2 per cent; of No. 5, it is 40.3 per cent (western, central and southern Illinois), and of No. 2 it is 42.3 per cent (northern Illinois).

T. A. Hendricks, Washington, D. C. (written discussion).—I collected from the West Virginia coal fields the samples that were used for this paper, I believe. In sampling these coals, I found it impossible in some instances to get lump samples of the required size from all parts of the bed. In other words, sometimes a part of the bed was so friable that 1½-in. lumps could not be broken out. I made several attempts to collect samples from pillars and found that this difficulty greatly increases in those cases.

H. G. Turner, State College, Pa.—Friability is probably more closely connected with moisture than has usually been thought, and should not be confused with hardness. The hardness of coal should be tested distinctly for its toughness, for it is apparent that a freshly-mined low-rank coal may be soft but tough. The condition and previous history of a sample have much to do with its friability; *e.g.*, lignite breaks down by drying but is quite tough when freshly mined. Shatter tests should be run at successive periods after mining to determine this characteristic, and tests should be run on freshly mined coal, pillar coal, and lumps taken in different ways—large, small, etc. In this way consistent friability data could be built up.

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[20] Amer. Soc. Mech. Engrs. (1931).

[21] C. M. Smith: The Friability of Illinois Coals. Univ. of Illinois Eng. Expt. Sta. *Bull.* 218 (1930).

G. H. Ashley, Harrisburg, Pa.—If the percentage of fixed carbon is multiplied by the British thermal units (in thousands), the resulting figures will separate coals of different ranks more than they were separated in the authors' charts.

A. C. Fieldner, Washington, D. C.—In regard to Dr. Cady's suggestion that Washington coals may not be representative of coals found elsewhere, more different kinds of coal are found there than in any other state, and the authors' tested coals from elsewhere give good comparisons. With the exception of the Illinois samples, all were taken at the face and immediately sealed.

H. F. Yancey and K. A. Johnson (written discussion).—Contrary to the opinions expressed by Mitchell and Smith in their discussion of the slacking results in our paper, the coals examined in this investigation were sampled by experienced engineers and geologists according to definite methods, and tested by carefully worked-out procedures. The samples of Washington coal were collected by S. H. Ash, district mining engineer, U. S. Bureau of Mines, who has had years of experience in Washington coal-mining operations. The West Virginia samples, as indicated in this discussion, were collected by T. A. Hendricks of the U. S. Geological Survey, and those from the Pennsylvania anthracite region by C. M. Stull, assistant mining engineer, and H. A. Goodman, scientific aid, U. S. Bureau of Mines. All samples were collected in the mine and sealed in tin-can containers for transmission to the laboratory except the one Illinois sample, which was courteously supplied by C. M. Smith, University of Illinois. This was an air-dried sample of sized coal from a Vermillion County mine used only in the friability and low-temperature assay tests.

The three coals from northern Illinois mentioned by Mitchell and Smith are not any that were examined in this investigation. These particular three coal samples are shown in the paper by Parr and Mitchell.[22]

Admittedly the slacking determination for coal is not a determination of great precision, owing to the nature of the test. Different layers of a coal bed may differ in physical properties, so it is necessary to use considerable care in selecting the sample for the slacking test. This was recognized in sampling the coals included in this investigation. In sampling the coal bed, particular pains were taken to include a proportionate amount of pieces from different layers comprising the bed, so as to make the sample representative of the bed at the place of sampling.

Considerable data are available on the limits of accuracy of the slacking test in a paper by Fieldner, Selvig and Frederic.[23] A comparison of the results obtained on duplicate samples from the same place in the mine shows that with careful sampling and testing the slacking index can be duplicated more closely than might be expected from a test of this nature. The experience of the Bureau of Mines with the test has shown it to be quite a reliable measure of a physical property of coal.

The friability tests were made on fresh coal; that is, coal containing all of its bed-moisture content. Only by proceeding in this manner may relative friability indexes for all ranks of coal be established. Obviously, the friability of low-rank coals increases on evaporation of its inherent or bed moisture, as has been demonstrated by unpublished trials made by the authors.

The friability of 12 per cent for Pennsylvania anthracite obtained by Smith and mentioned in Mitchell and Smith's discussion as being comparable to the authors' average friability of 33.2 per cent for 24 samples of anthracite from the eastern middle field of Pennsylvania was obtained by a drop method, whereas the authors' figure of

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[22] S. W. Parr and D. R. Mitchell: The Slacking of Coal and Its Proper Interpretation. *Ind. & Eng. Chem.* (1930) **22**, 1211.

[23] U. S. Bur. Mines *Rept. of Investigations* 3055.

33.2 per cent was obtained by a tumbler method. These two methods measure different specific properties or combinations thereof, which may not be directly related.

As stated in the text, the analyses of the coals reported in the paper, together with full descriptions of the methods used and the detailed results, are given in *Tech. Paper* 512, U. S. Bureau of Mines.

Some readers have requested further details of the method of calculating the friability index. The average or mean diameter or size before and after the test was calculated from the average areas of the sieve openings. For example, to obtain the average diameter of the sized product, the squares of the sieve openings through which the material passed and on which it was retained were added, this sum was divided by 2 and the square root was extracted. The weight of each sized product was multiplied by its average diameter and the sum divided by the total weight of all of the sized products.

Status of Scientific Classification of American Coals

By W. T. Thom, Jr.,* Princeton, N. J.

(New York Meeting, February, 1932)

Characteristics and Purpose of a Scientific Classification

Regarding the elements necessarily involved in working out a scientific scheme of classification, Stansfield and Sutherland,(94)† quoting Grout,(34) make the following statement:

> All bases (for a scientific classification) should be capable of quantitative determination, or be exactly related to some property capable of such a test, so that, if no natural groups are found, arbitrary lines may be drawn to limit the classes.
>
> The best basis for classification should involve inherent and fundamental qualities of the materials considered. Allied to this is the requirement that advantage should be taken of any natural grouping. As a further help in selection, the bases or tests should be easily applied, widely known, and have a wide range of values in the material classified.

In order that we may apply the philosophy thus expressed, it is necessary for us to have clearly in mind both the reasons why coals differ and the specific purposes to be served by the classification to be devised.

Coals differ because, as modern science has shown, coal beds are actually neither more nor less than buried and altered peat deposits, formed in remote geologic time. And as is generally known, such peat beds consisted of masses of partly macerated and decayed trees, shrubs, plants, mosses and/or algae, deposited under swamp or semiswamp conditions. It is therefore to be expected that coals will vary:

1. Because of differences in the parent vegetation, due either to evolutionary plant changes, as from the fernlike types of the Paleozoic to the coniferous and hardwood types of the late Mesozoic and Tertiary; or due to the differences in floral assemblages (growing at the same time) induced by differences in local environmental conditions. For example, moss, sedge, woody and other peats are today accumulating simultaneously in different bogs in North America.

2. Because of differences in the stage to which bacterial decay proceeded in the peat bed. Peats laid down in stagnant waters suffer

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† Superior figures in parentheses refer to a bibliography that is on file in the Engineering Societies Library, 29 West 39th St., New York, N. Y.

little additional decay after immersion, whereas peats formed in open water or in well drained swamps are subjected to prolonged bacterial action, which will destroy all except a few of the minor and more decay-resistant plant substances, such as resins, waxes, seed coatings, bark, etc.

3. Because of differences in the intensity of the pressure and heat which have affected the peat beds after burial, and because of differences in the length of time that the various coals have been subjected to such compression and heating.

Owing to the interplay of the above-mentioned factors, coals may exist in an almost endless variety of forms.

Moreover, as different coals have different use properties the classification of coals becomes a matter of practical importance as well as of great scientific interest. As we all know, many important modern industries are dependent upon the ready availability of coals possessing certain physical or chemical properties, and consequently the industrial world needs terms by which business men can specify the particular kind or kinds of coal they wish to purchase, and scientific men can describe coals needed for new processes and uses in terms of a scientifically exact and generally understood classification system.

EARLY HISTORY OF COAL CLASSIFICATIONS

The developing need for a coal classification was first indicated by the employment prior to 1800 of such terms as bitumen lapideum,(111) cannel coal,(51) Braunkohle,(46) lignite,(7) anthracite,(40) etc., and by the subsequent adoption of the terms semibituminous,(75) semianthracite,(75) and subbituminous coal.(10) Karsten,(47) Regnault,(72) Richardson,(73) Stein,(96) Fleck,(27) and Schondorff,(85) were other pioneers who proposed schemes for classifying foreign coals, and Johnson,(44) Rogers(75) and Frazer (30,31) were among the first to attempt to use chemical analyses as bases for a systematic classification of American coals, and Grüner,(38,39) Seyler,(80,81,82) Campbell,(8,9) Grout,(34) Parr,(63–65) Bauer,(4) Dowling,(18) White,(112) Ralston,(71) Ashley(1,2) and others have offered further proposals regarding systems for classifying coals.

SUGGESTIONS REGARDING BASIS TO BE USED FOR SCIENTIFIC COAL CLASSIFICATIONS

Mention was made in the previous paragraph of a number of the early coal classification systems put forward by different authors and comprehensive surveys of recent work along this line are contained in Fieldner's paper presented before the Second International Conference on Bituminous Coal in 1928;(21) in Stansfield and Sutherland's paper (1929) on the classification of Canadian coals;(94) in A. I. M. E. symposia of 1928 and 1930, and more recently by Bode(5) and Grumell.(37) Bases used in some of these classifications were as follows:

Regnault's Classification used proportions of chemical constituents shown in ultimate analyses.

Johnson's Classification applied to high-rank coals. It employed fuel ratio $\left(= \frac{\text{Fixed carbon}}{\text{Volatile matter}}\right)$ shown by proximate analyses of "dry, ash-free" coals.

Roger's Classification divided high-rank coals on percentage of volatile matter.

Frazer's Classification applied to high-rank coals. Used "fuel ratio" devised by Johnson.

Seyler's Classification was based on ratio of carbon to hydrogen in ultimate analyses of dry, ash-free coals.

Campbell's (early) Classification used carbon-hydrogen ratio of air-dried, ash-free coal.

Grout's Classification was based on proximate analyses of dry, ash-free coals. Used fixed carbon alone for anthracite coals, and fixed carbon and total carbon for lower rank coals.

Dowling's Classification employed the ratio

$$\frac{\text{Fixed carbon} + \frac{1}{2}\text{ volatile combustible}}{\text{Moisture} + \frac{1}{2}\text{ volatile combustible}}.$$

International Geological Congress Classification employed the ratio $\frac{\text{Fixed carbon}}{\text{Volatile matter}}$, flame characteristics and calorific value.

Ralston's Classification plotted carbon, hydrogen and oxygen percentages from ultimate analyses on a three-coordinate system.

Campbell's (later) Classification employed physical criteria and ratios of fixed carbon, volatile matter, and calorific values shown by proximate analyses of ash-free coals.

Parr's Classification employed volatile matter and calorific value of proximate analyses for dry, ash-free coal, in terms of "unit coal" as calculated by formula correcting for sulfur, etc.

Ashley's (Use) Classification was based on proximate analyses and physical factors, assigning also ash and sulfur limits determining "grade" of coal.

White's Classification classified according to position in metamorphic scale by classes and groups, and according to parent matter in common or "humic," splinty, canneloid and boghead types.

Fisher's(26½) *Classification* employed three coordinate systems, using fixed carbon, moisture and volatile matter from proximate analyses of as-received ash-free coal; or carbon, hydrogen and oxygen from ultimate analyses.

Stansfield's Classifications employed volatile matter, moisture content and calorific value on comparable (corrected) ash basis.

CURRENT STUDIES PRELIMINARY TO SCIENTIFIC COAL CLASSIFICATION

In order to consider Ashley's classification proposals made in 1923, the American Engineering Standards Committee (now the American Standards Association) called a meeting in Pittsburgh in November, 1926, and the existing system of committees on coal classification was consequently set up under the sponsorship of the American Society for Testing Materials. Fieldner(21) has described the make-up of these committees working on coal classification, and the preliminary results obtained by them have been summarized in A. I. M. E. symposia, of which the present one is the third.

Paleobotanic Studies of Coal

As early as 1778, Buffon and von Beroldingen had recognized the vegetal origin of coal and the *in situ* accumulation of this vegetation in ancient peat bogs(62) and abundant evidence confirming these views has accumulated during the nineteenth century. With the recent development of excellent techniques for studying coals in thin section under the microscope, or for studying them in polished (and etched) section, numerous workers in North America and Europe have greatly expanded our knowledge of coal-forming plant materials, and the bearing of this paleobotanic knowledge upon the problems of coal classification has been summarized by Thiessen,(99) who has described the evolutionary changes in the coal-forming flora during and since Carboniferous time, and has also given a summary of matters regarding plant chemistry and the chemistry of plant decay which bear directly on the genesis of coals.(98)

Concerning the progressive change in the coal-forming floras through geologic time Thiessen says:

> The representative plants during the Carboniferous were Calamites, Lepidodendrons, Sphenophylls, Cycadophytes, and Cordaites. The Permian marked a great transition period, so that in the Triassic great changes are noted, ... the age had become an age of gymnosperms. The Jurassic remained to be an age of gymnosperms. Among these, yews, cypresses, sequoias, cedars and pines were the characteristic trees . . . The Jurassic also saw the birth of the angiosperms, and by the Lower Cretaceous they were well on their way. Familiar plants were sassafras, laurel, myrtle, fig, aralia, oak, eucalyptus and palm . . . During the Upper Cretaceous the whole landscape attained a modern aspect. About 90 per cent of the plants were of the kinds known today. During the Tertiary the vegetation had assumed entirely modern aspects.

And with reference to the significance of these floral changes during geologic time in terms of coal character and coal classification, he says further "That the kind of plants contributing to a coal had a marked influence on the kind and type of coal produced can be easily shown."

Geologic Studies of Coal Origin and Metamorphism

The geological conditions under which coal swamps developed, and the nature of other conditions which brought about the progressive metamorphism of peat to anthracite and even to graphite, have been summarized by White,(113,114) who pointed out that variations in coal "type" result from variations in parent plant substances, and that the "rank" attained by a coal is a measure of both the heat and pressure the coal has been subjected to and of the time it has been subjected to such metamorphism. The importance of incipient rock metamorphism as the factor determining coal rank has also been further stressed by Campbell,(14) who concluded that the progressive metamorphism of a coal carries it successively through at least eight recognizable metamorphic stages or ranks. Geologists in general are in agreement with this view, and also with the idea that in so far as geologic evidence is concerned there is so complete a gradation between the various metamorphic ranks of coal that if definite limits between these ranks (or classes) of coal are to be set up, appeal must be made to supplementary chemical and physical tests to establish these limits.

One problem connected with coal classification arises from the fact that coal beds normally are not of uniform character and composition from top to bottom, but are made up of a series of layers formed of differing types of vegetal materials (which are notably unlike both in chemical composition and in purity). A single coal bed at a single locality may yield not one but several types of coals from different bands or layers composing the total thickness of the coal bed.

Geological and Chemical Evidence as to the Constitution of Coal

Individual coal-forming plant substances such as woody tissues, cuticles, resins, spore exines, etc., are highly complex chemically, as has been pointed out by Thiessen.(98) And as Fieldner and Davis(20) and others have also pointed out, these manifold coal-forming substances may be mingled in the most diverse proportions. Woody material impregnated with humic colloidal decomposition products is a major constituent of most coals and normally is characterized by brilliant black luster, and by cell structure visible under the microscope. Usage has not become standardized(101) but the terms "vitrain," "Glanzkohle," "vitrit," "provitrit," "anthraxylon," and by some, "clarain," have been used to designate this substance. "Mineral charcoal," "Mother of coal," "fusain," "fusit," or "Faserkohle" is another charcoal-like substance commonly forming thin layers in coal beds. And the main "groundmass" or "attritus," in which the woody layers are commonly embedded consists, according to Thiessen,(102) of woody degradation matter, degradation matter from other carbohydrate tissues, spores and pollens, cuticles resins, oil algae and/or opaque matter. Certain "opaque matter," which is the predominant constituent of some coals, gives these coals the matte surface, "dull" color, and hardness which are regarded as characteristic of "splint coal," "durain," "durit" or "Mattkohle."

Certain broad chemical and physical differences exist between the "vitrain," "fusain" and "durain" of any coal bed,(37) but as each of these substances almost always occurs intimately mixed with varying proportions of the other two, their recognition serves to illustrate the complexity of the coal classification problem rather than to afford an easy or direct approach to the establishment of a satisfactory classification system.

Physicochemical and Physical Properties of Coals

In considering the constitution of coal, it is to be noted that the lower rank coals contain large percentages of water, due to their colloidal character,(55) and that the properties of these coals are in large measure due to the presence of this moisture.(5) To eliminate completely such water it is "necessary to heat coal to about 300° C. with a continual removal of water vapor,"(41) and as this carries the coal above temperatures at which appreciable quantities of volatile constituents, other than water, are liberated from the coal,(70) attempt to classify low-rank coals on a "dry" basis is not feasible.(104)

Various physical properties of coals (which are indirect expressions of their chemical or physicochemical composition and metamorphic stage) are color, fracture or jointing, friability, density, chemical reactivity and resistance to weathering. Anthracites, for example, are black and lustrous—massive in character—and break with a characteristic conchoidal fracture like massive glass. Bituminous coals are characterized by rectangular jointing; commonly show many alternating bands of bright and dull coal; and the higher rank bituminous coals show little or no deterioration on weathering. Lignites, on the other hand, are brown to dark brown in color; commonly show visible woody texture; have an irregular fracture; and rapidly "slack" and break down when subjected to weathering. An interesting and effective method for comparing these and other classifications based respectively on proximate and ultimate analytical data has been devised and described by Rose,(78) and by the use of his chart the position of a coal in one classification can be closely estimated in terms of another classification.

Problems of Coal Analysis Bearing on Scientific Classification of Coal

From the foregoing facts it becomes obvious that the development of a scientific classification of coals is dependent upon the availability and use of accurate and comparable analytical data regarding the constitution and properties of coal. The procurement of such accurate and comparable data involves the selection of satisfactory mine samples; the analysis of these samples by standard methods and under closely controlled and standard laboratory conditions; the conversion of analytical results to a comparable ash basis; the employment of the corrected analyses for classification purposes according to sound scientific principles, and a correct scientific understanding of the problems dealt with.

Sampling.—Only the analyses of unweathered coals afford true and comparable guides to the composition and relative rank of coals. Therefore fresh face samples are needed and samples from weathered ribs and outcrops should be excluded when coals are being classified. The composition of different coal layers in the same face shows notable differences, and these layers may vary in number and relative thickness along the strike, therefore the average of several face samples of the same bed is the best guide to the time average composition and character of a particular coal.(37) The detailed procedure to be followed in sampling and analysis of coals has been well summarized by Moore(62) and need not be repeated here.

Chemical and Physical Analysis.—The methods to be used in coal analysis for classification purposes were also reviewed by Fieldner in 1930(22) and by Grumell in 1931(37) and these descriptions were coupled with estimates as to the "influence of variations in method on published analytical results" as contained in reports issued by the Bureau of Mines. Table 1 (p. 208), showing permissible variations in analytical results under standard American methods, was also given.

The existence of relations between rank or property differences of coals and corresponding (measurable) chemical differences in the coals is indicated by such quantitative determinations as have been made of (1) oxidizability;(28,29,61,67,79,87) (2) slacking properties;(25,120) (3) agglutinating properties;(60,120) friability;(120) and (4) alkali-soluble ulmin content(93) of coals—all of these determinations tending to confirm a close natural relationship between the chemical character, physical properties and water content of the coals studied.

Methods for determining the relation between character of coals and the volume and composition of the gaseous compounds volatilized during their distinctive distillation have been suggested by the work of Lebeau(52) and of Dolch and Gieseler,(17) and some data on the organic sulfur compounds and nitrogenous constituents which may be characteristic of different coals have been supplied by Jolly and Wheeler,(45) and by Cobb.(15) These sulfur compounds, if present in appreciable quantities, undoubtedly affect the character of a coal, and thus affect its classification, as has been pointed out by various men conversant with industrial practices.

Mahadevan's work has also suggested the possibility that X-ray analysis of coals can be used to obtain indication of their constitutional

TABLE 1.—*Normal Tolerances between Determinations*

A. On Laboratory Samples, Crushed to Pass through an 840-Micron (No. 20) Sieve.	
Moisture:	
Under 5 per cent	0.3
Over 5 per cent	0.5
B. On Laboratory Samples, Crushed to Pass through a 250-Micron (No. 60) Sieve.	
1. Moisture:	
Under 5 per cent	0.3
Over 5 per cent	0.5
2. Ash:	
No carbonates present	0.3
Carbonates present	0.5
Coals with more than 12 per cent of ash, containing carbonate and pyrite	1.0
3. Volatile matter:	
Bituminous coals	1.0
Lignites	2.0
Cokes (high temperature)	0.4
4. Sulfur:	
Coal, under 2 per cent	0.10
Coal, over 2 per cent	0.20
Coke	0.05
5. Ultimate analysis (author's estimate):	
Carbon	0.4
Hydrogen	0.1
Nitrogen	0.1
6. Calorimetric determination:	
Permissible differences, per cent	0.5
7. Fusibility of coal ash:	
Permissible differences	50° C.

differences as well as information regarding the form and distribution of their contained ashy impurities.(56,59)

Ash Determination and "Correction." The problem of the accurate quantitative determination of the ash-forming mineral matter in raw coal is one of the important ones to be dealt with in any classification scheme involving a comparison of either proximate or ultimate analyses. Several types of mineral matter may occur in coals(48,6) among which are:

1. Finely disseminated matter (especially in bright coals) which is mineral matter that was contained by the original coal-forming vegetation.

2. Layers and flakes, mainly on bedding planes, consisting of clayey matter deposited contemporaneously with the coal.

3. Shale particles mingled with the coal, or crystals and masses of infiltered calcite, ankerite, pyrite, marcasite (and more rarely gypsum) filling joint cracks, or infiltered into coal, especially into fusain layers. Some constituents of the ash-forming mineral substances lose weight upon heating because of the evolution of water, carbon dioxide, etc.; and others may gain weight by oxidation. And to complicate matters further, the range in percentages of ash in the fusain, durain and clarain-vitrain fractions of a typical coal bed may be, respectively, from 15.6 to 6.3 and 1.2 per cent.(53)

The history of efforts to perfect ash-correction formulas was given in 1930 by Fieldner and Selvig,(24) and by Stansfield and Sutherland,(95) and a further discussion of this topic by Fieldner, Selvig and Gibson was published this year.(26)

DECISIONS REACHED REGARDING SCIENTIFIC CLASSIFICATION OF AMERICAN COALS

In determining the present status of the scientific classification of American coals, four phases of the problem must be considered: (1) The basis of classification to be used; (2) the general categories of coals to be recognized in the proposed classification system; (3) the names or designations to be used in referring to the groups, classes and types of coals recognized under phase 2, and (4) the tests to be used in determining the correct classification of "border-line" coals.

Basis of Classification to Be Used

It has been agreed that the primary basis for the scientific classification of American coals should be "evolutionary rank," which is indicated by the degree of metamorphic dehydration, devolatilization and jointing or consolidation exhibited by the coals.

It has also been agreed that coals are to be classified on the basis of their composition as they occur in nature, that is to say, on the as-received (ash-free) basis, with "bed" moisture included (where such moisture is of appreciable amount), for, as Ashley has said, "Any (comprehensive coal) classification that starts out by rejecting the moisture of coal can be neither scientific nor practical."[1]

It has been tentatively agreed that fixed carbon and British thermal units, determined under standard methods of proximate analysis and compared on the same ash basis, form the best criteria available for establishing the framework for a scientific classification for North American coals. The B.t.u. value of coal is regarded as the most significant single criterion by which coals of dissimilar properties can be separated, and fixed carbon 100 per cent (volatile matter + moisture) is the next most satisfactory guide in classification. The B.t.u. and fixed-carbon values used are to be provided by proximate analyses, made under standard laboratory conditions. The samples are to be unweathered, and are to contain not over 10 per cent ash and 2.5 sulfur; and several (probably at least five) samples are to be analyzed and considered before a coal of debatable character is to be classified.

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[1] When dealing only with coals of very high rank, it may, however, be expedient or convenient to use a subsidiary classification based on dry, ash-free analyses.

The "comparable ash basis" to be used in classification will probably be obtained by calculating the as-received proximate analysis to the ash-free form. Whether or not various corrections are to be applied in the conversion to the ash-free form is still under consideration.

General Categories of Coals to Be Recognized in Proposed Classification System

Coals are to be divided into "groups" and "classes" according to the stage of their metamorphic alteration in their transportation from peat to anthracite, and into "types" on the basis of the predominant parent vegetal constituents, according to decisions of the Committee on Scientific Classification.

According to suggestions made by Stansfield and Sutherland(94) (October, 1929) and resolutions adopted by the A. S. T. M. Committee on Scientific Classification (Nov. 22, 1929), coals are to be divided into four groups or classes, namely: (1) anthracitic group, (2) bituminous group,[2] (3) subbituminous group,[2] and (4) lignite group.

In February, 1930, Campbell suggested an adaptation and elaboration of this plan of subdivision, as follows:

"Brown coal" group (moisture above 27 per cent)	1. Peat class 2. German brown coal class 3. Lignite class
Hydrobituminous group (moisture 27 to 5 or 6 per cent)[3]	1. High-moisture (western subbituminous) class 2. Low-moisture (Mississippi Valley high-moisture coal) class
Bituminous group (moisture less than 5 or 6 per cent)	1. High-volatile class 2. Low-volatile class
Anthracite group	1. High-volatile class 2. Low-volatile class 3. Meta-anthracite class

However, the committee has not yet passed upon the question of substituting this classification scheme of Campbell's for the one just previously described.

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[2] Should be divided into three—upper, middle and lower—classes, according to Stansfield and Sutherland.

[3] Cady suggested that if such a classification is to be used, the lower moisture limit for this group should be set at 9.5 or 10 per cent instead of at 5 or 6 per cent.

On March 26, 1930, the committee voted to recognize three classes each in the anthracitic and lignitic groups as follows:

Group	Class (Typified by)
Anthracitic	1. Rhode Island "anthracites" (meta-anthracites, subgraphitic) coals 2. Normal Pennsylvania anthracites 3. "Bernice anthracites" (semianthracites, subanthracites)
Lignitic	1. Lignites (brown lignites) 2. Low-rank or unconsolidated "brown coals" 3. Peats

At the same meeting several men expressed their belief that the coal classes to be recognized next below anthracitic rank should be:

Class	Volatile Matter
Low-volatile	Up to not over 20 to 22 per cent
Medium-volatile	20 or 22 to about 30 per cent
High-volatile (low moisture)	Above 30 per cent
High-volatile (high moisture)	Illinois type

In 1930 it was also suggested that coals be divided into four "types" on the basis of differences in their parent vegetal materials, these types being: (1) "common, banded" or "humic" coals; (2) splinty coals; (3) cannelloid coals; and (4) algal coals. Of these types the humic and splinty coals normally show a notably banded structure on vertical joint surfaces, and the cannelloid and algal coals are not banded.

Also it has been tentatively suggested that coals be divided into attrital, attrital-anthraxylous, anthraxylous-attrital and anthraxylous, on the basis of the relative proportions of contained woody (anthraxylous) and of (attrital) "groundmass" materials.

Names or Designations to Be Given Groups, Classes and Types of Coals to Be Recognized in Proposed Classification System

Various suggestions have been made as to the precise scientific names to be adopted for the various subdivisions of the projected coal classification system, but practically no attention has been given this subject as yet, and the suitability of the descriptive terms which have been applied to coal groups, classes and types in the foregoing pages is yet to be determined, when the nature of the classification to be proposed is finally worked out.

Tests to Be Used in Determining Classification of "Border-line" Coals

At the present time attention is being largely centered upon the development of tests which will give both accurate and accurately reproducible quantitative information as to the chemical composition and physical properties characteristic of individual coals. This is necessary to meet business and legal demands as well as scientific requirements, and involves matters of standard analytical practice; ash correction and calculation; sulfur and nitrogen determination, and the like. Much more exact information is needed regarding the physical properties of coals, and the causes and significance of variations in these properties. Current studies of agglutinating value, friability and slacking are aimed at the ascertainment of how the physical properties of coals relate to their chemical composition, and as to what guides to natural groupings of coals their physical differences may afford. Possibly X-ray studies of coals may afford significant evidence as to variations in internal structure of different ranks and types of coal, and analytical determination of the total amount and relative percentages of various gases evolved from coals when subjected to destructive distillation (under standard conditions) may also afford a helpful clue to the correct classification of certain coals.

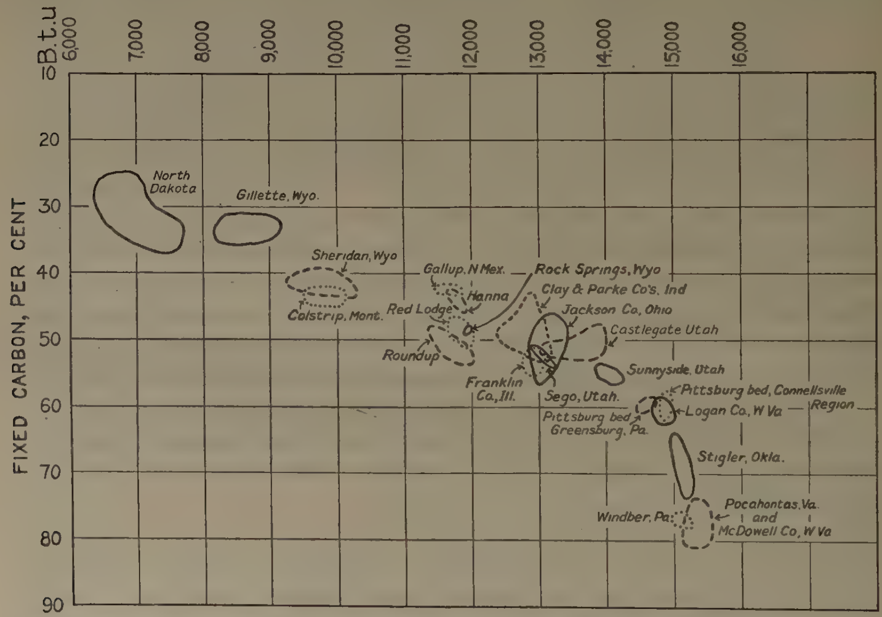

Fig. 1.—Comparison of typical American coals on basis of British thermal units and fixed carbon, on as-received, ash-free form of proximate analyses.

Graphic Study of Coal Analyses as a Guide to Further Work Needed to Perfect a Scientific Coal Classification

A series of graphic studies of proximate coal analyses computed to the as-received, ash-free basis, have been carried forward by the writer, assisted by Mrs. L. E. Stevens, and the results of this work to date are believed to confirm the view that fixed carbon and calorific values of coal, determined on this basis, do afford a sound basis for the scientific classification of North American coals. As an illustration of this, the writer has prepared Fig. 1, which shows the relative position on fixed carbon-B.t.u. coordinates of 25 coals, selected by Campbell and Ashley as geologically typical coals, covering the metamorphic range from lignitic to semibituminous coals.

It is probable that some refinement in ash computation taking into account sulfur and perhaps other constituents may improve the relative spacing of certain coals on this system of coordinates, and doubtless index numbers for friability, slacking and the like must be considered in setting group and class boundaries, but there seems to be no reason to doubt that a graphic classification method employing B.t.u. and fixed carbon as coordinates does work best in bringing different coals into rational systematic relationships with each other.

Summary and Conclusion

Tests by which group, class and type limits can be accurately and definitely fixed remain to be worked out, but sufficient progress has been made to indicate that the B.t.u. value and fixed carbon content of coals as they occur in nature will be the essential criteria on which a scientific coal classification must be based.

It has been decided that coals are to be divided into four major groups of classes (anthracitic, bituminous, subbituminous and lignitic), and that classes within both the anthracitic and lignitic groups are to be recognized, while at least five and probably six subdivisions of the bituminous-subbituminous groups are to be set up. Four types of coals—woody or "humic," splinty, cannelloid, and algal or boghead—are probably to be recognized.

It is probable that when the final comprehensive classification is set up it will be published with parallel statements showing the degrees of correspondence between it and other systems now in use (such as the Seyler and Parr systems) which are being currently applied to coals within restricted ranges of composition and character.

DISCUSSION

(*H. J. Rose presiding*)

W. T. Thom, Jr.—The present study was confined to certain coals recognized as types, and was made to see whether unlike types were separated by the scheme of classification used.

H. J. Rose, Pittsburgh, Pa.—Considering coals from Sunnyside, Utah, and Franklin County, Illinois, in regard to coking properties, the Utah coal is higher in hydrogen, but otherwise they are similar and have almost indistinguishable coking characteristics. Presumably the Utah coal lies above the water table while the Franklin County coal is from a deep vein. This may explain the difference of 5 to

6 per cent in mine moisture. It would be desirable to have these coals tested by the Stansfield method to determine moisture at 100 per cent relative humidity.

G. S. Rice, Washington, D. C.—The Sunnyside coal, which is a coking coal in the great Book Cliff field of Utah, dips below the water table. The coking area of the field occupies a relatively small area, and its coking quality changes in character on either side of the Sunnyside (Utah Fuel Co.) and Columbia Steel Company's mines.

H. N. Eavenson, Pittsburgh, Pa.—There are considerable differences in the properties of Pocahontas coal and coal from McDowell County, West Virginia.

R. D. Hall, New York, N. Y.—It is a mistake to talk about the Bernice field as producing a single rank of coal, as some true anthracite is present.

W. T. Thom, Jr.—The analyses represent a composite of the entire field.

Member.—I question the practice of basing classification on ash-free coal, inasmuch as coal has a decided content of inherent ash from its source materials, so that ash-free coal is not, in reality, coal at all.

H. J. Rose.—This is an important point. A scientific classification compares coals on the basis of their actual chemical and physical properties. However, it is desirable for some purposes to compare coals on a mineral-matter-free basis. A committee is soon to report on the feasibility of using float coal as a basis for classification.

W. T. Thom, Jr.—Microscopic and X-ray studies of coal have revealed three kinds of ash: (1) initial or inherent ash, finely disseminated, less than 1 per cent; (2) particles and flakes intimately mixed with the coal; (3) shale partings and joint plane infiltrations. As a rule the last group is the largest. It is also the most variable.

H. J. Rose.—The inherent ash and even the total ash in coal is frequently much less than would be calculated from the source plant materials.

W. T. Thom, Jr.—Wood ash is somewhat soluble, with the result that growing plants use it over, fresh growths leaching out the old ash from the peat bed in which the plants are rooted.

H. J. Rose.—A number of schemes for coal classification based on an arbitrary ash content have been proposed, but none have proved quite satisfactory.

Proposed Method for Determining the Oxidation Temperature of Anthracite

By J. Leland Myer,* Bethlehem, Pa.

(New York Meeting, February, 1932)

Some of the early experiments on the oxidation temperature of coal[1] were undertaken in England in connection with a study of self-heating, or spontaneous combustion. It was then suspected that coals with low oxidation temperature were most likely to ignite by self-heating, but further experiments showed that many additional factors were involved.

The oxidation temperature has importance as a physical property or characteristic of the coal. While only a few of the physical properties have been studied, they appear to have basic significance[2] in the ultimate classification of all coals. When reliable data become available on the oxidation temperatures, as determined by standard and adequate methods, it should provide an additional basis for classification and comparative evaluation.

Oxidation Temperature Defined

For such a heterogeneous substance as coal is known to be, no simple definition of oxidation temperature can be established on a theoretical basis. If coals were homogeneous and of simpler structure, the accepted definition of "ignition temperature" as applied to simple gaseous mixtures might be adopted; that is, the temperature at which self-heating of the mixture sets in. But most coals self-heat slightly even at storage temperature, as evidenced by spontaneous combustion, and the definition for gases cannot be applied to coals with any exactness.

By subjecting coals to suitable experimental conditions various oxidation characteristics can be obtained and examined for critical or significant stages of the oxidation process. Oxidation temperature itself must then be defined on the basis of such experimental results.

Experimental Methods

Attempts to determine critical points in the temperature functions of coal undergoing oxidation have resulted in the following experimental methods:

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* Research Fellow, Department of Physics, Lehigh University.

[1] Coal Dust Expts. Committee Work, Mining Assn. of Great Britain (1909). Unpublished.

[2] H. F. Yancey and K. A. Johnson. See page 171.

1. *Disappearance of Oxygen in a Closed System.*—Wheeler[3] has described a method whereby a column of pulverized coal is heated in a closed system, and the enclosed air or oxygen is circulated continuously through the coal column. Periodic measurements are made on the rate of disappearance of oxygen. While such determinations give no information on a definite oxidation temperature, they do indicate the rate of oxygen combination with the coal substance under the various temperature conditions. Porter and Ralston[4] have used this method successfully to study the early stages of the oxidation, chiefly with bituminous coal.

2. *The Glow Point.*—This is proposed by Arms[5] as a new criterion in the study of oxidation temperatures. The glow point is the temperature at which a glow of the coal mass first becomes visible. While the test is very simple, the importance of its results is held somewhat in question.

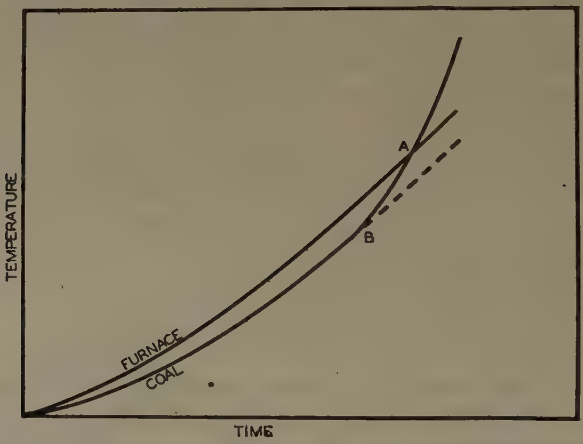

Fig. 1.—Method for determining critical oxidation temperature by intersection of time-temperature curves.

3. *Carbon Dioxide Index.*—This scheme, proposed by Parr and Coons,[6] involves the measurement of the temperature at which carbon dioxide is first detected in the gases escaping from heated coal. The carbon dioxide index has had considerable application in the study of lignite.[7] While it does not reveal all that may be desired, the results appear definite and reproducible.

4. *Intersection of Time-temperature Curves.*—Wheeler[8] has also described this method, in which temperatures are recorded at regular

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[3] R. V. Wheeler: *Jnl.* Chem. Soc. (1918) **113,** 945.

[4] H. C. Porter and O. C. Ralston: U. S. Bur. Mines *Tech. Paper* 65. (1914).

[5] R. W. Arms: Univ. of Ill. Engr. Expt. Sta. *Bull.* 128 (1922).

[6] S. W. Parr and C. C. Coons: *Ind. & Eng. Chem.* (1925) **17,** 118.

[7] W. C. Eaton, G. A. Brady, A. W. Gauger, I. Lavine and C. A. Mann: *Ind. & Eng. Chem.* (1931) **23,** 87.

[8] Reference of footnote 3.

intervals, both for the coal specimen being heated and for the surrounding furnace wall. Fig. 1 shows a typical plot of such data. The temperature at the intersection of the curves (point *A* in the diagram) is defined as the critical oxidation temperature. While the temperature at the intersection of the curves is certainly a critical point in the oxidation process of the coal, some experimenters attribute greater importance to the point where uniform lag between the coal and furnace temperature ceases[9] (*B* in the diagram). It is here that the coal first acquires heat from its own combustion at a fairly rapid rate. But this point cannot be determined accurately. Even the critical point *A* has been found to vary widely in a series of determinations.

The General Motors combustibility test for coke[10] is a modification of this method, the furnace heating curve being dispensed with entirely. A sudden temperature rise in the vicinity of the coke sample is taken as an indication of its ignition. Exact test conditions are maintained, this being essential for the reliability of results.

5. *Temperature-resistivity Curves—a Proposed Method.*—For anthracite coals, a definite break in the slope of the temperature-resistivity curve has been found. The corresponding temperature is that at which rapid oxidation might reasonably be expected to take place. From this and other considerations the point appears significant, and will warrant a closer study of the details attendant.

APPARATUS FOR THE TEMPERATURE-RESISTIVITY METHOD

The electrical resistance of anthracite had been studied previously in connection with other physical properties[11] and a satisfactory type of resistivity cell had been devised. For the present investigation this cell was modified (1) for use inside an electric furnace, (2) for taking temperature measurements of the coal mass, and (3) for the constant circulation of gas through porous electrodes and the coal mass.

The details of the apparatus are shown in Fig. 2. The charge of coal consists of a 2-gram mass of anthracite "as received" and pulverized to pass through a 20-mesh screen on to a 40-mesh screen. This coal is contained in a Pyrex glass tube, ¾-in. inside diameter, mounted vertically inside an electric furnace. Above and below the coal mass are pieces of fine metal gauze which fit tightly into porous metal plug electrodes. Wires attached to these electrodes enable resistance measurements to be made on the coal mass by means of an external bridge circuit (a Wheatstone bridge with multiplier device, not shown here).

Porcelain tubes prevent too great a heat transfer from the furnace and electrodes into the metal thrust rods at top and bottom. The lower thrust rod enters the Pyrex tube through a rubber stopper, and is kept cool by a water bath at its base. The upper rod carries the load platform and weights (total 4 kg.) for holding the coal under constant pressure and in good contact with the electrodes. Coal temperatures are measured with a thermojunction which extends up through the lower electrode.

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[9] J. D. Davis and D. A. Reynolds: U. S. Bur. Mines *Tech. Paper* 409 (1928).

[10] A. L. Boegehold: *Trans.* Amer. Foundrymen's Assn. (1929) **37**, 91.

[11] J. L. Myer: A. I. M. E. *Tech. Pub.* 482 (1932).

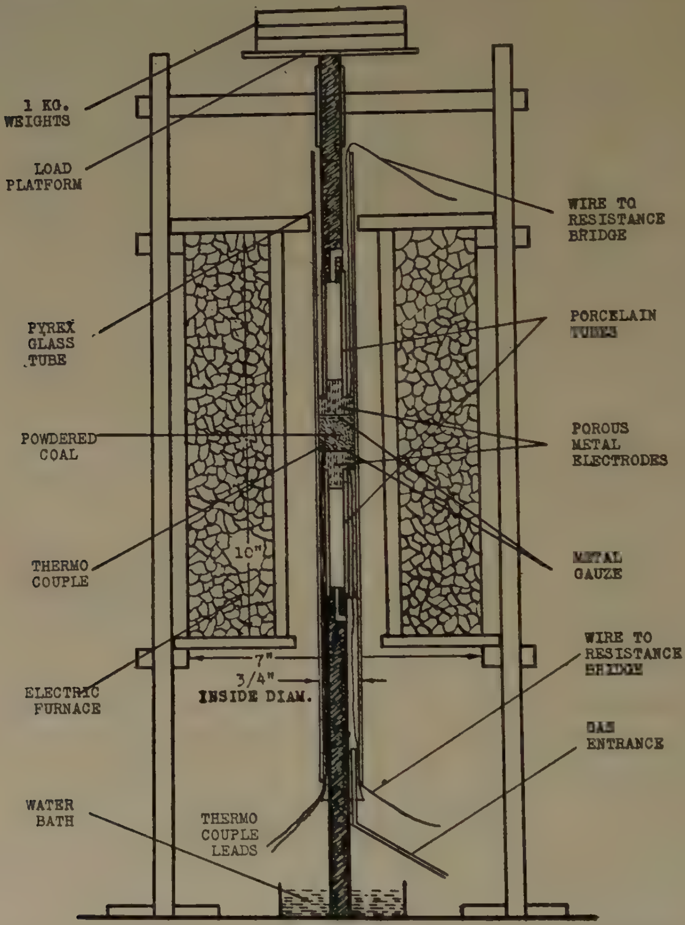

FIG. 2.—CROSS-SECTION OF MODIFIED RESISTIVITY CELL AND FURNACE.

Gas enters the glass tube through the stopper in its base. It is preheated as it passes upward through the porous electrode into the coal mass and escapes at the top of the glass tube. The gases used were commercial oxygen and nitrogen, dried in a calcium chloride tower before entering the resistivity cell, and admitted separately (for comparative tests) at the rate of 180 c.c. per second throughout the time of heating.

After preliminary trials, these conditions were found to give satisfactory results. The typical test consisted of passing dry gas through the coal sample for five minutes, and then raising the furnace temperature constantly at the rate of 8° C. per minute. The resistance bridge was kept balanced continuously, and temperature and resistance readings recorded every minute. Numerous short runs were made to determine the degree of oxidation at specific temperatures.

INTERPRETATION OF THE TEMPERATURE-RESISTIVITY CURVES

Typical data are shown for two samples of anthracite; from Brookside colliery, Orchard vein in Figs. 3 and 4 and from Bear Valley colliery, top split of Mammoth vein in Figs. 5 and 6. The complete range of temperature-resistivity measurements is shown in Figs. 3 and 5. On this scale it is not possible to include the individual data points. There is no evidence of an abrupt change of slope in any of the curves.

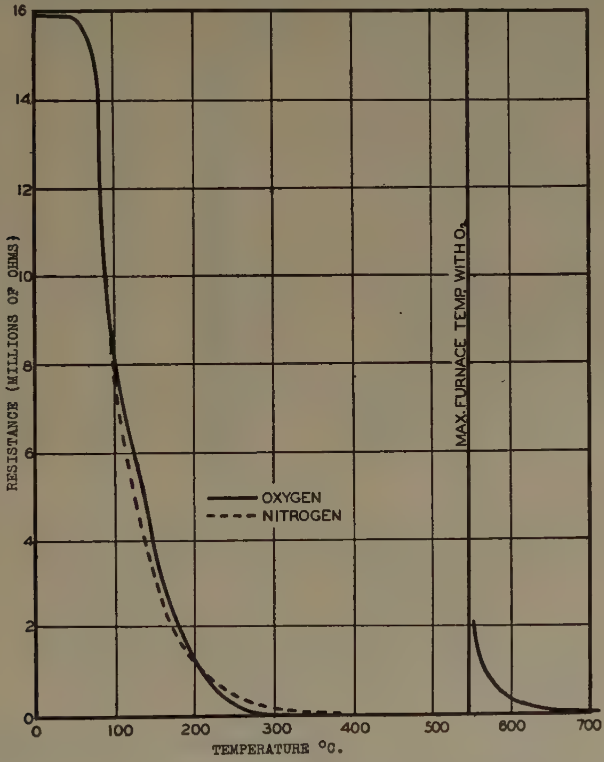

FIG. 3.—TEMPERATURE-RESISTANCE CURVE, BROOKSIDE COLLIERY, ORCHARD VEIN.

Figs. 4 and 6 contain the interesting portions of the curves at the higher temperatures, and with the resistance scale enlarged 100 times. Here also the individual data points (taken at 1-min. intervals, except full black points) can be shown, and the distinct change of slope of the curves when oxygen is used becomes outstanding.

The items to be considered in the interpretation of these curves are the following:

1. Superposition of the curves at the outset. As might be anticipated, the nature of the gas has no effect on the resistance variation at low temperatures. The coincidence of the curves may be taken as an indication of the reproducibility of resistance measurements with the apparatus used.

2. The sustained fall of resistance in the 100° C. region. Previous tests[12] by the author indicate that this marked change of resistance is intimately connected with the loss of moisture of the coal. Following standard practice for drying, some coals lost as much as 45 per cent while others gained upward to 30 per cent of their initial moist resistance when dried. Prolonged exposure to the atmosphere restored nearly all the original resistance to the specimens.

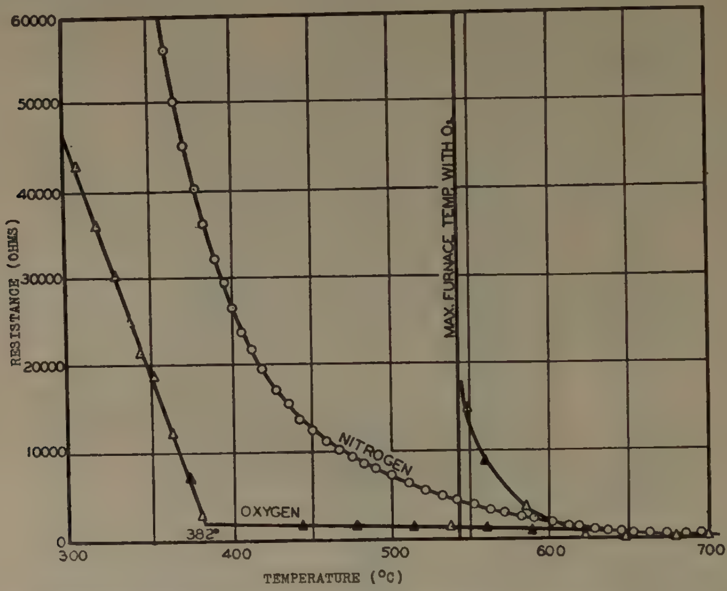

Fig. 4.—Temperature-resistance curve, Brookside colliery, Orchard vein. Data taken at 1-mm. intervals except full black points.

3. Gradual divergence of the oxygen and nitrogen curves. The resistance change from loss of moisture may be considerably prolonged beyond the 100° C. region, but eventually the curves diverge, and here must arise the earliest effects of oxidation. The mechanism which connects this resistance change with the early stages of oxidation appears to be very obscure. Porter and Ralston[13] found the formation of

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[12] Reference of footnote 11.

[13] Reference of footnote 4.

unstable compounds or complexes of oxygen with part of the coal mass first taking place in the oxidation process.

4. Thermal agitation from external and internal heating. Continuous heating of the furnace, combustion of the volatile matter, and eventually of the solid matter must create considerable electronic activity in the coal mass. Such electronic activity has been studied, and is known to have important effects in the electrical properties of flames and incandescent bodies.[14] Combustion is also gradually removing some of the solid matter, diminishing the conducting mass between the electrodes.

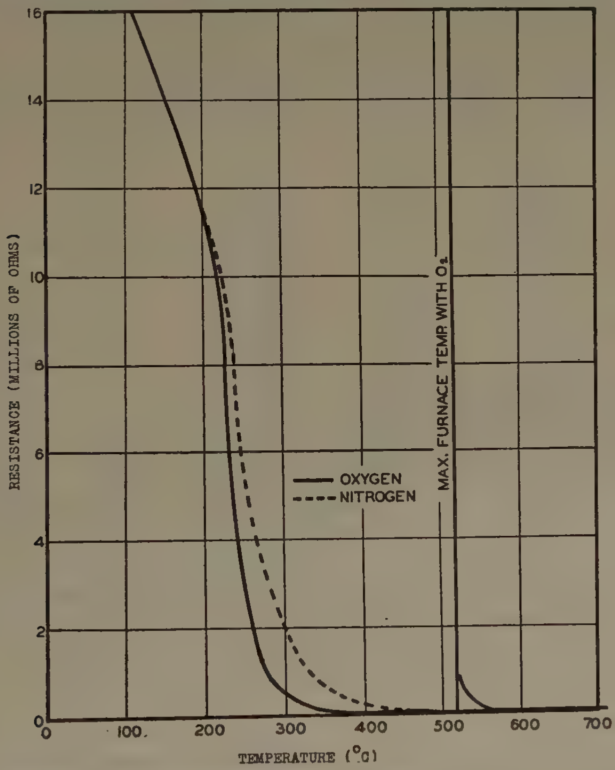

FIG. 5.—TEMPERATURE-RESISTANCE CURVE, BEAR VALLEY COLLIERY, TOP SPLIT OF MAMMOTH VEIN.

5. Formation of ash and the final high resistance values. While all the factors enumerated are tending to decrease the resistance at a rapid rate, the ash being formed is a poor conductor and increases the resistance. At the distinct break in the curve the formation of ash may set in rapidly and prevent a much lower resistance drop. And when finally the process of oxidation is complete, the resistance assumes incredibly high values because ash is the sole conductor.

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[14] Wilson: Electrical Properties of Flames and Incandescent Bodies. Univ. of London Press, 1912.

6. The rapidity of oxidation. Inspection of the curves will show the rapid rise of temperature following the abrupt change of slope on the oxygen curve. In 2 or 3 min., the coal temperature exceeds that of the furnace by several hundred degrees. The specimen temperature then slowly decreases, approaching the maximum furnace temperature. At this point, when the coal has completely turned to ash, the resistance suddenly rises so rapidly that it is impossible to balance the bridge circuit for any readings.

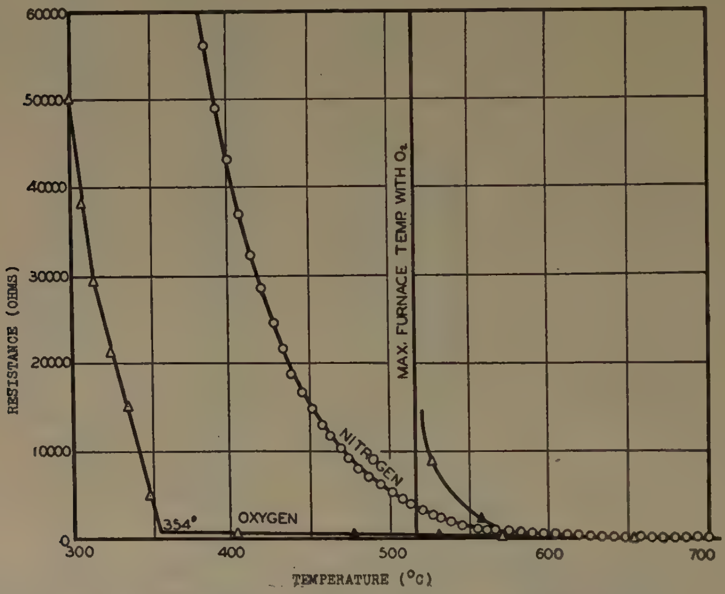

FIG. 6.—TEMPERATURE-RESISTANCE CURVE, BEAR VALLEY COLLIERY, TOP SPLIT OF MAMMOTH VEIN. DATA TAKEN AT 1-MM. INTERVALS, EXCEPT FULL BLACK POINTS.

7. Smooth curves obtained with nitrogen. Even when nitrogen gas is used, some evidence of critical points might reasonably be expected, but the curves give no such indications. The resistance drop is uniformly steady, finally resulting in the low values (of the order of a few ohms) previously obtained for carbonized coal.[15] A much slower rate of furnace heating might reveal that the temperature-resistivity curves with nitrogen are not regular but contain critical points.

GENERAL CONCLUSIONS

The point where the resistance slopes change abruptly is believed to be of basic significance, and might reasonably be established as a new criterion for the determination of oxidation temperatures. Coal samples examined just before reaching this point show no visible oxidation of the solid constituent. The critical point of these resistance measurements seems closely related to the glow point, as a visible glow is easily noted to follow the region of this critical point. How it is related to other oxidation-temperature determinations is not known, but this would certainly provide an interesting study.

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[15] Reference of footnote 11.

At present a limited number of anthracite samples have been used for the experiments. The results obtained appear consistent, and a series of successive determinations is given in Table 1. It cannot be stated what success the method may have in immediate application to other classes of coal. Certain modifications may be desirable, as well as a more thorough understanding of the factors responsible for the resistance changes noted. Possibly a slower rate of heating, as previously suggested, would aid materially in such a study.

TABLE 1.—*Successive Determinations of Oxidation Temperature by the Temperature-resistivity Method*

	Brookside Colliery, Orchard Vein, Deg. C.	Bear Valley Colliery, Top Split of Mammoth, Deg. C.
	385	351
	382	359
	381	355
	383	354
	381	357
	387	353
Average	383	355

The outstanding feature to be emphasized is the simplicity of the test, and the easy control of the important variables. Likewise the reproducibility of results is encouraging, and may enable further correlations in the classification and comparative evaluation of coal.

Application of Ash Corrections to Analyses of Various Coals*

By A. C. Fieldner,† Washington, D. C., W. A. Selvig‡ and F. H. Gibson,§
Pittsburgh, Pa.

(New York Meeting, February, 1932)

A former paper[1] described in detail various methods of calculating coal analyses to obtain the composition and calorific value of the pure coal substance—that is, of the coal free from its mineral matter. The paper included a table showing calorific values of pure coal, as calculated from float-and-sink portions, by means of the Parr formula for "unit coal" and by the customary moisture-free and ash-free formula. A modification of the Parr formula was proposed, based on the assumption that one-half the sulfur present in coal occurs as pyritic sulfur and one-half as organic sulfur. In the published discussion of the paper Cady gave a table of results on a large number of Illinois coals as obtained by the Parr formula and the proposed modification thereof. The results obtained by Cady indicate that the Parr formula applies better to Illinois coals than does the modified formula proposed by Fieldner and Selvig.

The present paper gives results obtained by applying the modified Parr formula to the analyses of the float-and-sink portions of the coals shown in the former paper, and results obtained by applying the Parr formula, the modified Parr formula, and the simple moisture-free and ash-free formula to various coals of the United States ranging in sulfur content from less than 1 per cent to more than 4 per cent sulfur. A graphic method of estimating the calorific value of pure coal and determining the relation of the coal ash to the mineral matter of coal, as proposed by Stansfield and Sutherland,[2] was applied to a number of coals, and the results are given herein.

Testing Formulas with Float-and-sink Separations

The former paper by Fieldner and Selvig gave heat values of pure coal of the float-and-sink portions of 21 coals, as calculated by the Parr formula for "unit coal." The float-and-sink separations were made in a solution of benzene and carbon tetrachloride adjusted to a specific gravity of 1.35. This method of testing the formulas assumes that the pure coal of the float portion and of the sink portion are identical in composition and that the calculated calorific values of the two portions should be the same, provided the formula used is correct. With the coals tested the float portions contained from 2.0 to 4.5 per cent ash and the sink portions from 16.0 to 35.9 per cent ash. Table 1 shows the variations of calorific value of the pure coal of the sink portions as compared to that of the float portions, as calculated by the simple moisture-free and ash-free formula, the Parr formula, and the modified Parr formula. The formulas as applied to the moisture-free analyses are as follows:

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* Published by permission of the Director, U. S. Bureau of Mines.

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§ Assistant Chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

[1] A. C. Fieldner and W. A. Selvig: Present Status of Ash Corrections in Coal Analysis. *Trans.* A. I. M. E., Coal Div. (1930) 597–613.

[2] E. Stansfield and J. W. Sutherland: Determination of Mineral Matter in Coal and Fractionation Studies of Coal. *Trans.* A. I. M. E., Coal Div. (1930) 614–626.

$$\text{Moisture-free and ash-free B.t.u.} = \frac{\text{Indicated B.t.u.}}{1.00 - \text{ash}}$$

$$\text{Parr “unit coal” B.t.u.} = \frac{\text{Indicated B.t.u.} - 5000\ \text{sulfur}}{1.00 - (1.08\ \text{ash} + \frac{22}{40}\ \text{sulfur})}$$

$$\text{Modified Parr “unit coal” B.t.u.} = \frac{\text{Indicated B.t.u.} - 4650\ \text{sulfur}}{1.00 - (1.08\ \text{ash} + \frac{3}{4}\ \text{sulfur})}$$

The heating values of the pure coal of the high-ash sink portions as calculated by the simple moisture-free and ash-free formula are obviously too low. Better agreement between the heating values of the pure coal of the float-and-sink portions is shown for the Parr formula and the modified Parr formula. Neither of these two formulas, however, gives good agreement for some of the coals, as shown by maximum differences of 500 B.t.u. for the Parr formula and 850 B.t.u. for the modified formula. These differences show that the variation in the nature of the mineral matter in different coals makes it difficult to devise any mathematical formula that will apply universally and that considerable errors may result in applying either of these formulas to high-ash coals. The average arithmetical difference for the Parr formula is 175 B.t.u. and for the modified Parr formula is 201 B.t.u. For 12 of the 21 coals the Parr formula gives the best agreement.

Fish and Addlestone[3] made similar float-and-sink separations on samples from 10 Virginia coal beds. They applied the three formulas to the analyses of the untreated coals and of the float-and-sink portions. The average of differences between the lowest and highest B.t.u. obtained for the coals tested was 442 for the moisture-free and ash-free formula, 175 for the Parr formula, and 156 for the modified Parr formula. The 10 coals tested ranged from 2.2 to 19.1 per cent ash and from 0.46 to 1.17 per cent sulfur, on the moisture-free basis. Carbon dioxide present as inorganic carbonate was determined on the untreated coals and on the

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[3] F. H. Fish and J. A. Addlestone: Unit Coal Studies on Some Virginia Coals. *Ind. & Eng. Chem.*, Analytical Ed. (1931) **3**, 155–158.

TABLE 1.—*Variation of B.t.u. of Pure Coal of Sink Portions as Compared to That of Float Portions*

Coal Designation	State	County	Condition	Analysis, Dry-coal Basis		Difference of B.t.u. of Pure Coal of Sink Portions as Compared to That of Float Portions		
				Ash, Per Cent	Sulfur, Per Cent	Moisture and Ash-free	Unit Coal by Parr Formula	Unit Coal by Modified Parr Formula
A	Pennsylvania	Somerset	Float	2.9	0.77	− 780	−330	−270
			Sink	26.0	1.69			
B	Maryland	Allegany	Float	4.3	0.86	− 100	+100	+110
			Sink	16.0	1.08			
C	Pennsylvania	Clearfield	Float	2.9	0.87	− 900	−190	+ 70
			Sink	29.2	5.25			
D	Pennsylvania	Somerset	Float	3.5	0.62	− 530	− 50	+140
			Sink	21.5	4.53			
E	Pennsylvania	Cambria	Float	2.5	1.02	− 590	− 10	+180
			Sink	25.0	4.59			
F	Ohio	Meigs	Float	4.0	1.08	− 590	+280	+580
			Sink	35.9	5.75			
G	Illinois	Williamson	Float	3.7	2.41	− 410	+500	+850
			Sink	34.6	7.38			
H	Pennsylvania	Westmoreland	Float	4.0	1.0	− 630	− 40	+ 90
			Sink	29.3	3.3			
HW	Pennsylvania	Westmoreland	Float	4.3	1.2	− 540	−110	+ 20
			Sink	23.2	3.6			
I	West Virginia	(New River Coal)	Float	2.0	0.7	− 590	−210	−140
			Sink	22.0	2.0			
J	West Virginia	(Pocahontas Coal)	Float	2.1	0.6	− 410	− 90	− 70
			Sink	20.9	0.8			
K	Ohio	Jefferson	Float	4.4	1.5	− 570	− 30	+170
			Sink	25.7	5.1			
L	West Virginia	Raleigh	Float	2.0	0.7	− 570	−150	−110
			Sink	24.0	1.5			
M	Pennsylvania	Mercer	Float	3.5	0.9	− 730	−170	− 70
			Sink	28.7	2.7			
N	Illinois	Williamson	Float	4.5	1.8	− 300	+170	+270
			Sink	26.8	3.3			
O	Pennsylvania	Westmoreland	Float	4.0	1.1	− 900	− 80	+120
			Sink	35.8	4.2			
P	Pennsylvania	Allegheny	Float	3.8	1.0	− 530	+150	+260
			Sink	33.0	2.8			
Q	Pennsylvania	Westmoreland	Float	3.6	0.8	− 350	+ 30	+ 80
			Sink	24.5	1.3			
R	Pennsylvania	Somerset	Float	3.3	0.5	−1050	−350	− 50
			Sink	27.4	6.1			
RW	Pennsylvania	Somerset	Float	3.5	0.5	− 790	−230	0
			Sink	24.0	5.0			
S	Illinois	(Mixture from 7 mines)	Float	4.5	2.6	− 290	+410	+580
			Sink	32.9	4.7			
		Average arithmetical difference				579	175	201
		Average algebraic difference				− 579	− 19	+134

float-and-sink portions. It was found that some of the coals contained appreciable amounts of carbonates as the carbon dioxide content of the untreated coals ranged from 0.03 to 1.41 per cent. The carbon dioxide was added to the ash in case of the Parr formula and the modified Parr formula and much better agreement between the unit B.t.u. values was obtained. The average difference was then found to be 82 B.t.u. for the Parr formula and 96 B.t.u. for the modified Parr formula. Evidently, for this group of coals, an additional correction of ash for carbon dioxide gives satisfactory agreement in unit coal B.t.u.

Testing Formulas by Analyses of Samples from the Same Mine

The calorific value of pure coal from the same bed in restricted areas should be fairly constant. The formulas were tested on analyses of bed samples of coal from the same mine, and the mines selected were those from which a number of analyses were available with a wide range in ash content. As the only difference between the Parr formula and the modified formula consists in the corrections applied to the sulfur, the mines selected covered a wide range of sulfur content. The coal analyses were taken from analyses made and published by the Bureau of Mines. The mines were grouped according to sulfur content, and coals from 10 different mines are included under each group. The grouping according to approximate sulfur content is shown in Table 2.

Table 2.—*Coals Grouped According to Sulfur Content*

Group	Approximate Sulfur Content of Dry Coal, Per Cent	Number of Mines	Samples per Mine	Total Samples
1	Less than 1........	10	5	50
2	1 to 2.............	10	5	50
3	2 to 3.............	8	5	40
		2	4	8
4	3 to 4.............	8	5	40
		2	4	8
5	More than 4.......	3	5	15
		1	4	4
		6	3	18
	Total...........	50		233

Table 3 summarizes the results of the calculations of calorific values of pure coal for each group of mines. The calorific values were calculated for the individual samples from each mine, and the maximum variation was recorded for each mine. The average variation of the individual samples from the mean values was also determined for each of the mines. These values were then averaged for the 10 mines of each group. Table 3 shows comparatively little difference in the results as obtained by the three formulas. Probably this is due to the fact that the variations in ash of individual samples from the same mine were not sufficiently great to detect inaccuracies in the formulas used for calculating calorific values of pure coal. The average differences obtained, even with the simple moisture-free and ash-free formula, were practically the same as for the two formulas which apply corrections. The regular Parr formula by this test gave slightly better results for each group of coals than did the modified Parr formula.

Table 3.—*Average Variations of Calculated B.t.u. of Pure Coal for All Mines of Each Group of Mines*

Group	Moisture and Ash-free B.t.u.		Unit Coal, B.t.u.			
			Parr Formula		Modified Parr Formula	
	Average Maximum Variation	Average Variation from Mean	Average Maximum Variation	Average Variation from Mean	Average Maximum Variation	Average Variation from Mean
1	140	40	129	40	136	43
2	173	55	141	39	147	39
3	133	45	149	47	179	55
4	109	36	105	32	123	39
5	157	46	103	39	143	54
Average	142	44	125	39	146	46

Fish and Addlestone made similar tests on 28 different samples of coal from the Merrimac bed of Virginia. These samples varied in ash from 7.4 to 44.2 per cent, in sulfur from 0.32 to 3.71 per cent, and in carbon dioxide from 0.26 to 5.40 per cent. Without correction for carbon dioxide the average deviation from the mean of all the samples was 192 for the Parr formula and 179 for the modified Parr formula. On correcting for carbon dioxide the average deviation from the average was 75 for the Parr formula and 86 B.t.u. for the modified Parr formula. Here again correction for carbon dioxide gives better agreement and has been shown to be necessary for coals containing appreciable amounts of inorganic carbonates.

Testing Formulas by Graphic Method

A third test was made on the formulas for pure coal by a direct comparison with the graphic method for estimation of calorific value of pure coal as proposed by Stansfield and Sutherland.[4] For the graphic-method test the coal was ground to pass a 20-mesh sieve and 50 to 100-gram portions were placed in mixtures of benzene and carbon tetrachloride adjusted to the following specific gravities: 1.30, 1.33, 1.35 and 1.38.

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[4] Reference of footnote 2.

After standing about 2 hr. at constant temperature the float-and-sink portions were separated, and the excess liquor was removed by filtration under suction. The portions were then placed in vacuum desiccators containing paraffin shavings and kept under a vacuum of about 20 mm.

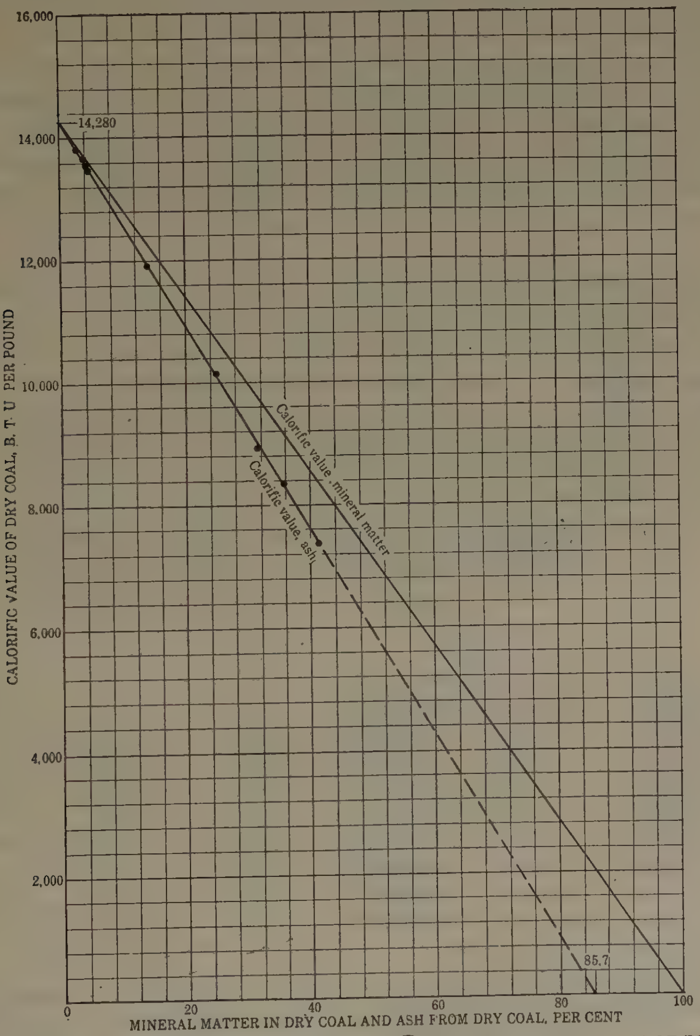

FIG. 1.—GRAPHIC METHOD FOR ESTIMATION OF BRITISH THERMAL UNITS OF PURE COAL AND RELATION OF ASH TO MINERAL MATTER.

of mercury for about 24 hr. to remove the carbon tetrachloride and benzene. At the end of that time only a faint odor of these chemicals could be detected, indicating that only negligible amounts were present in the coal. The float-and-sink portions were then rapidly crushed to pass a 60-mesh sieve and analyzed for moisture, ash, sulfur and calorific value. The calorific values for all the samples were determined in the same calorimeter.

Coals from seven mines were tested by the graphic method. To illustrate the method of plotting, the results obtained for one of the coals is shown in Fig. 1. The analysis of this coal and of the various float-and-sink portions is given in Table 4. The figure gives the moisture-free calorific values plotted against the ash percentages of the original coal and of the various float-and-sink portions. The points practically all fall on a straight line, and the calorific value of the mineral-matter free coal, as determined by extrapolation, is 14,280 B.t.u. For this particular coal the heating value calculated by means of the Parr formula for "unit coal" is 14,175 B.t.u., and by the modified Parr formula is 14,328 B.t.u.

Fig. 1 shows that the calorific-value ash curve meets the line of zero calorific value at 85.7. The ratio of the mineral matter to ash for this coal is then 100 to 85.7, or 1.17. This ratio includes all corrections necessary to apply to the ash of this coal to correct for loss of water of hydration, presence of iron pyrites, carbonates, and other constituents that may be volatile or change in weight in the ash determination.

TABLE 4.—*Analyses of Untreated Coal and of Float-and-sink Portions of Sample from No.* 6 *Bed, Williamson County, Illinois*

Sample	Laboratory Number	Moisture-free Coal		
		Ash, Per Cent	Sulfur, Per Cent	B.t.u.
Untreated coal	A-65835	14.03	3.94	11,917
Float on 1.30	A-65836	2.86	2.46	13,801
Float on 1.33	A-65838	3.90	2.49	13,653
Float on 1.35	A-65840	4.16	2.58	13,591
Float on 1.38	A-65842	4.69	2.61	13,491
Sink in 1.30	A-65837	24.85	5.28	10,156
Sink in 1.33	A-65839	31.40	6.27	8,925
Sink in 1.35	A-65841	35.78	6.80	8,343
Sink in 1.38	A-65843	41.41	6.77	7,374

Table 5 shows the results obtained with the graphic method for coals from seven mines. For five of these mines only single samples were available, for one mine two samples are shown, and for one mine five samples were tested. Assuming that the B.t.u. values of the pure coal as determined by the graphic method are correct, the variations from these as shown for the Parr formula and the modified Parr formula represent the errors of calculating "unit coal" heating values for these coals. These differences indicate that there is apparently no choice between these two formulas, as the average differences between the computed results as

TABLE. 5.—*Comparison of Calorific Values of Pure Coal as Determined by Graphic Method and by Formulas for "Unit Coal"*

State	County	Bed	Laboratory Number	Dry Coal Analysis		Calorific Value of Pure Coal, B.t.u.				Ratio of Mineral Matter to Ash by Graphic Method
				Ash, Per Cent	Sulfur, Per Cent	Graphic Method	Difference of Calculated Values from Graphic Values: Moisture and Ash-free	Parr Formula	Modified Parr Formula	
Alabama	Jefferson	Mary Lee	A-65555	14.68	1.04	15650	−357	−100	− 57	1.15
Alabama	Walker	Black Creek	A-68738	5.79	0.97	15020	− 89	+ 18	+ 53	1.11
	(2 samples from same mine)		A-68819	12.40	2.49	15120	−241	+ 25	+125	1.13
Illinois	Peoria	No. 5	A-63429	14.02	2.94	14525	−413	−125	− 10	1.12
Illinois	Williamson	No. 6	A-65835	14.03	3.94	14280	−419	−105	+ 48	1.17
Ohio	Jefferson	Pittsburgh No. 8	A-65940	8.80	2.29	15000	−318	−124	− 39	1.22
Pennsylvania	Allegheny	Thick Freeport	A-65368	8.10	0.96	15320	−134	+ 9	+ 45	1.10
Pennsylvania	Allegheny (5 samples from same mine)	Thick Freeport	A-68865	10.27	1.34	15320	−204	− 12	+ 40	1.10
			A-68866	7.82	1.49	15370	−192	− 33	+ 23	1.12
			A-68867	7.80	1.55	15310	−156	+ 4	+ 63	1.12
			A-68868	6.07	1.23	15290	−157	− 33	+ 12	1.11
			A-68869	7.10	2.13	15310	−145	+ 28	+109	1.12
				Average arithmetical difference			235	51	52	
				Average algebraic difference			−235	− 37	+ 34	

compared to those obtained by the graphic method are of the same order of magnitude. The ratio of mineral matter to ash as obtained with the graphic method ranges from 1.10 to 1.22. For the mine from which five samples were tested this ratio is fairly constant, as it ranges from 1.10 to 1.12.

The graphic method for arriving at the composition of pure coal has the disadvantage of the labor involved in the float-and-sink separations and the number of analyses required. For low-ash coals it is probable that sufficient accuracy for coal classification purposes can be obtained by the application of empirical formulas to the analyses of the untreated coals. For high-ash coals, however, the errors inherent in such formulas may give relatively large errors in the estimated composition of the pure coal, and for such coals it would be best to reduce the mineral matter materially before making the analyses. This could be done conveniently by a float-and-sink separation on a solution of 1.35 or 1.38 sp. gr. Such float portions were available for the coals used in the graphic method; so a comparison was made of the calculated calorific values of the pure coal of these float portions with the calorific values of the pure coal as obtained by the graphic method.

Table 6 shows the percentages of ash and sulfur in the float portions, as obtained by using solutions of 1.35 and 1.38 sp. gr. The percentages of the total coal recovered in the float portions is also shown. The coal recovered in the float on the 1.35 sp. gr. solution ranged from 36.8 to 87.4 per cent of the total coal, and that recovered on the 1.38 sp. gr. solution ranged from 57.6 to 89.9 per cent. The ash in the 1.35 sp. gr. float portions ranged from 1.45 to 4.96 per cent, and that in the 1.38 sp. gr. float portions from 1.69 to 5.78 per cent. In general the sulfur content in the float portions of the two different gravity solutions was practically the same. The relatively large amounts of sulfur remaining in the float portions of the two Illinois coals show the difficulty of removing this constituent from some coals. This is probably due to the presence of finely disseminated pyrite. Sulfur forms were not determined in the float portions, but undoubtedly a considerable portion of the sulfur was present as organic sulfur, a form of sulfur that cannot be removed by float-and-sink methods.

Table 7 shows the variation of calculated calorific values of the pure coal of the float portions from the calorific values of pure coal as obtained by the graphic method previously described. The Parr formula as applied to the float portions shows the best agreement for most of the coals. With the exception of one coal the values obtained for the Parr formula on the coals floated on the 1.35 sp. gr. solution are within 50 B.t.u. of the calorific values obtained by means of the graphic method. The best agreement, however, was obtained in applying the Parr formula to the coal floated on the 1.38 sp. gr. solution, as the variations between

TABLE 6.—*Ash and Sulfur Content of Coals Floated on Solutions of 1.35 and 1.38 Specific Gravity*

State	County	Bed	Original Coal, Moisture-free		Float on 1.35 Sp. Gr.			Float on 1.38 Sp. Gr.		
			Ash, Per Cent	Sulfur, Per Cent	Float Portion, Per Cent of Total Coal	Ash, Per Cent of Moisture-free Coal	Sulfur, Per Cent of Moisture-free Coal	Float Portion, Per Cent of Total Coal	Ash, Per Cent of Moisture-free Coal	Sulfur, Per Cent of Moisture-free Coal
Alabama..........	Jefferson	Mary Lee	14.68	1.04	62.7	4.96	0.81	71.0	5.78	0.82
Alabama..........	Walker	Black Creek	5.79	0.97	87.4	1.45	0.94	89.5	1.69	0.86
	(2 samples from same mine)		12.40	2.49	76.4	1.75	1.04	78.3	2.00	1.12
Illinois............	Peoria	No. 5	14.02	2.94	36.8	3.76	2.15	57.6	5.45	2.17
Illinois............	Williamson	No. 6	14.03	3.94	69.1	4.16	2.58	74.3	4.69	2.61
Ohio..............	Jefferson	Pittsburgh No. 8	8.80	2.29	76.4	4.16	1.46	82.3	4.62	1.55
Pennsylvania......	Allegheny	Thick Freeport	8.10	0.96	81.4	3.25	0.66	85.1	3.77	0.65
Pennsylvania......	Allegheny	Thick Freeport	10.27	1.34	79.3	3.30	0.66	82.5	3.62	0.66
	(5 samples from same mine)		7.82	1.49	82.1	3.19	0.74	86.0	3.70	0.75
			7.80	1.55	81.5	3.09	0.63	84.9	3.45	0.70
			6.07	1.23	86.6	2.98	0.56	89.9	3.35	0.57
			7.10	2.13	81.2	2.96	0.96	85.6	3.23	0.98

TABLE 7.—*Variations of Calorific Values of Pure Coal of Float Portions on Solutions of* 1.35 *and* 1.38 *Sp. Gr. from Calorific Values of Pure Coal as Obtained by the Graphic Method*

State	County	Bed	Calorific Value of Pure Coal as Obtained by Graphic Method	Differences of Calculated Calorific Values of Float Portions from Graphic Values, B.t.u.					
				Float on 1.35 Sp. Gr.			Float on 1.38 Sp. Gr.		
				Moisture and Ash-free	Parr Formula	Modified Parr Formula	Moisture and Ash-free	Parr Formula	Modified Parr Formula
Alabama................	Jefferson	Mary Lee	15,650	− 66	+ 31	+ 61	−118	−10	+20
Alabama................	Walker	Black Creek	15,020	− 30	+ 18	+ 51	− 33	+17	+46
	(2 samples from same mine)		15,120	− 18	+ 39	+ 75	− 71	− 7	+31
Illinois................	Peoria	No. 5	14,525	+ 26	+141	+215	−144	− 9	+68
Illinois................	Williamson	No. 6	14,280	− 99	+ 28	+116	−125	+ 9	+99
Ohio................	Jefferson	Pittsburgh No. 8	15,000	−150	− 49	+ 2	−151	−40	+15
Pennsylvania............	Allegheny	Thick Freeport	15,320	− 42	+ 22	+ 46	− 46	+25	+49
Pennsylvania............	Allegheny	Thick Freeport	15,320	− 68	− 4	+ 20	− 82	−12	+12
	(5 samples from same mine)		15,370	− 72	− 5	+ 21	− 66	+ 9	+35
			15,310	− 43	+ 18	+ 41	− 71	− 3	+22
			15,290	− 46	+ 10	+ 30	− 43	+21	+41
			15,310	− 89	− 17	+ 17	− 50	+26	+60
		Average arithmetical difference		62	32	58	83	16	42
		Average algebraic difference		− 58	+ 19	+ 58	− 83	+ 2	+42

these results and those by the graphic method were within 50 B.t.u. for all of the coals. This is good agreement and indicates that satisfactory analyses of pure coal can be obtained by a float-and-sink separation on a solution of 1.38 sp. gr. and the application of corrected ash formulas to the analyses of the float portions. Probably it is not necessary to reduce the ash content of high-ash coals below 6 per cent, and this was accomplished for all the coals tested by means of the 1.38 sp. gr. solution. An advantage of using the higher gravity solution is that less coal is discarded in the sink portions. To keep the ash in the float portions below 6 per cent it may be necessary for some coals to make the separation in a solution of 1.35 specific gravity.

Conclusions

The Parr formula for "unit coal," which assumes that practically all the sulfur in coal is present as pyritic sulfur, and the modified formula proposed by Fieldner and Selvig, which assumes that one-half the sulfur is in the form of pyritic sulfur, were subjected to tests on various coals by (1) float-and-sink methods, (2) computations on analyses of coals with varying ash content, from the same mine, and (3) comparison with the Stansfield and Sutherland graphic method. Although the assumption that one-half the sulfur in coal is present as pyritic sulfur is approximately correct for most coals, the tests applied and described in this paper fail to show that the modified formula has any advantage over the regular Parr formula for "unit coal." Both formulas require additional correction for carbon dioxide if carbonates are present in appreciable amount in the coal. The graphic method for estimating the heating value of pure coal and determining the ratio of mineral matter to ash gave satisfactory results when applied to a number of coals. Close agreement with the graphic method was obtained by float-and-sink separations on a solution of 1.38 sp. gr. and applying the Parr formula to the analyses of the float portions.

DISCUSSION

(H. J. Rose presiding)

G. H. Cady, Urbana, Ill. (written discussion*).—In view of the conclusions reached by the authors, the results of somewhat similar studies of Illinois coals may be of interest. The investigation consisted, (1) in the comparison of values for mineral matter in coal derived by the application of empirical formulas to items of the proximate analysis with values derived by calculation from analyses of the ash, and (2) in the comparison of the results obtained by applying the different formulas to a group of 32 analyses of No. 6 coal samples in various mines in Macoupin County, Illinois.

The formulas used for the derivation of mineral matter from the items of the proximate analysis are shown on the following page:

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* Presented with the permission of the Chief, State Geological Survey, Urbana, Ill.

1. Mineral matter = 1.08 ash + 0.55 sulfur
2. Mineral matter = 1.08 (ash + CO_2) + 0.55 sulfur
3. Mineral matter = 1.08 ash + 0.75 sulfur
4. Mineral matter = 1.08 (ash + CO_2) + 0.75 sulfur
5. Mineral matter = 1.08 ash + 0.25 sulfur
6. Mineral matter = 1.08 ash + 0.8 CO_2 + 0.25 sulfur
7. Mineral matter = 1.08 ash + 0.4 sulfur
8. Mineral matter = 1.08 ash + 0.5 sulfur
9. Mineral matter = 1.1 ash + 0.5 sulfur
10. Mineral matter = 1.1 ash + 0.4 sulfur

The two first formulas are those proposed by Parr and Wheeler;[5] the third and fourth are modifications of Parr and Wheeler's formulas proposed by Fieldner and Selvig.[6] The fifth formula was proposed by Cady in 1931;[7] the sixth is a modification of the fifth and separates the carbonates in the ash from the clay-forming constituents. The seventh represents an intermediate position between the first and the fifth, and is not mathematically developed. After comparisons, using values derived by each of the first six formulas, it seemed probable that formula 7 would give more satisfactory results and it was accordingly tried. Formula 8 was suggested by Dr. Fieldner in recent conversation as an improvement over formula 1. Further simplification of the Parr-Wheeler formula was suggested in the recent perusal of a report on Alberta coals by Stansfield, Hollies and Campbell,[8] and this simplification is indicated by formula 9. Trial of this formula indicates the advisability of trial of formula 10.

The calculation of mineral matter from the ash analysis was based on determinations of the amount of the following constituents in the ash: SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, Cl, alkalis as Na_2O and SO_3, leaving an undetermined balance. The mineral matter was calculated by using these values and the values for pyritic sulfur and CO_2 derived from mineral matter, as follows: The amount of pyrite was derived by calculation from the value for pyritic sulfur. Magnesium carbonate (dolomite) was determined by calculation from MgO; the balance of CO_2 was used to determine the amount of calcium carbonate present. The balance of CaO was calculated as gypsum. In one or two instances the amount of Fe_2O_3 and the amount of CO_2 was large, and the two were combined to form iron carbonate. Al_2O_3 and SiO_2 were combined to form clay and any excess SiO_2 was regarded as quartz. Therefore the minerals present were regarded as pyrite, iron oxide, dolomite, calcite, gypsum, clay and quartz. The amount of chlorine and alkali being very small in both instances, they were ignored. The undetermined balance was regarded as mineral matter. The total mineral matter consisted in the sum of the values for the various minerals listed plus the undetermined balance. Since the empirical formulas in several instances assume that the mineral matter includes also the organic sulfur, two values for total mineral matter were derived, one without organic sulfur and one with organic sulfur. A tabulation of these data is available in blueprint form.

The matter of interest is the comparison of the values derived by calculation from the ash analyses and the values derived by the use of empirical formulas (Table 8). Values derived by using formula 1 should theoretically be compared with values

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[5] S. A. Parr and W. F. Wheeler: Unit Coal and the Composition of Coal Ash. Univ. of Illinois, Eng. Expt. Sta. *Bull.* 37, (1909).

[6] A. C. Fieldner and W. A. Selvig: Reference of footnote 1.

[7] G. H. Cady: Discussion of reference 2.

[8] E. Stansfield, R. T. Hollies and W. P. Campbell: Analyses of Alberta Coal Scientific and Industrial Research Council of Alberta *Rept.* 14 (1925) 17.

TABLE 8.—*Deviation of Values Determined by Empirical Formulas from Mineral Matter Values Calculated from Ash Analyses*

PART 1.—COMPARISON OF VALUES IN WHICH ORGANIC SULFUR IS INCLUDED AS PART OF THE MINERAL MATTER

Kind	Formula 1 Deviation		Formula 2 Deviation		Formula 3 Deviation		Formula 4 Deviation	
	Units	Cases	Units	Cases	Units	Cases	Units	Cases
Positive	0.34	2	6.49	14	4.06	11	23.77	39
Negative	63.05	58	31.69	45	28.64	49	8.63	20
Agreement		0		1		0		1
Total	63.39	60	38.18	60	32.70	60	32.40	60
Average positive	0.17		0.46		0.37		0.61	
Average negative	1.09		0.70		0.58		0.43	
Average total	1.06		0.64		0.55		0.54	
Maximum positive	0.22		1.27		0.49		1.78	
Maximum negative	3.38		1.73		2.66		1.01	

PART 2.—COMPARISON OF VALUES IN WHICH ORGANIC SULFUR IS NOT INCLUDED AS PART OF THE MINERAL MATTER

	Formula 1 Deviation		Formula 5 Deviation		Formula 6 Deviation		Formula 7 Deviation		Formula 8 Deviation		Formula 9 Deviation		Formula 10 Deviation	
	Units	Cases	Units	Cases	Units	Cases	Units	Cases	Units	Cases	Units	Cases	Units	Cases
Positive	30.21	45	0.71	4	7.95	23	8.29	24	21.34	38	32.65	45	16.32	36
Negative	7.32	15	42.37	56	20.23	36	19.71	35	11.10	22	7.45	15	11.44	24
Agreement		0		0		1		1		0		0		0
Total	37.53	60	43.08	60	28.18	60	28.00	60	32.44	60	40.10	60	27.76	60
Average positive	0.49		0.18		0.35		0.35		0.51		0.73		0.45	
Average negative	0.67		0.76		0.[illegible]6		0.56		0.56		0.50		0.48	
Average total	0.63		0.72		0.47		0.47		0.54		0.67		0.46	
Maximum positive	1.81		0.33		0.84		1.02		1.55		1.83		1.30	
Maximum negative	1.49		2.61		1.38		2.07		1.73		1.48		1.75	

which include organic sulfur. Inspection of the data shows, however, that they are much nearer the values which do not include organic sulfur. Values derived by using formula 1 are therefore included in both Part 1 and 2 of the table. Values derived by using formulas 2, 3 and 4 are much nearer those derived from the ash with the organic sulfur included than those in which the organic sulfur is not included. It does not seem necessary, therefore, to include these in Part 2.

The summary (Table 8) indicates definitely that certain formulas provide values in much closer agreement with the values derived by calculation from the ash analysis than do others. Choice among the formulas should be determined by the amount of deviation, the formula giving the least deviation presumably being the most desirable. In most particulars formula 10 displays the most desirable characteristics. Total deviation is least, positive and negative deviations, although not least, are most nearly equal and average deviations are most nearly equal, average total deviation being least. The maximum deviations provide little basis for discrimination.

Interest attaches particularly to formulas 1, 7, 8 and 10. Formula 1 gives values which are slightly too high; formula 8 gives values slightly lower than the values given by formula 1 but still slightly too high. Formula 7 gives values prevailingly too low. The best formula appears to lie between formula 8 and formula 7, which is the position occupied by formula 10.

Formulas 1, 2, 3 and 4 do not in any case give values as close in agreement with the values in which organic sulfur is included (Part 1, Table 1) as do formulas 1, 5, 6, 7, 8, 9 and 10, with respect to values in which the organic sulfur is not included (Part 2, Table 1). In general, formulas that make allowance for CO_2 are not particularly satisfactory.

The comparison of the unit coal values obtained by applying the 10 formulas to a set of 32 analyses of the same coal in a relatively small area can be summarized briefly.

The formulas used for the determination of the "unit" or mineral-free coal are the following:

11. Unit coal $= \dfrac{\text{B.t.u.} - 500\text{ S}}{1.00 - (1.08 + 0.55\text{ S})}$ (Parr)

12. Unit coal $= \dfrac{\text{B.t.u.} - 5000\text{ S}}{1.00 - (1.08\ (\text{ash} + CO_2) + 0.55\text{ S})}$ (Parr)

13. Unit coal $= \dfrac{\text{B.t.u.} - 4650\text{ S}}{1.00 - (1.08\text{ ash} + 0.75\text{ S})}$ (Fieldner)

14. Unit coal $= \dfrac{\text{B.t.u.} - 4650\text{ S}}{1.00 - (1.08(\text{ash} + CO_2) + 0.75\text{ S})}$ (Fieldner)

15. Unit coal $= \dfrac{\text{B.t.u.} - 2624\text{ S}}{1.00 - (1.08\text{ ash} + 0.25\text{ S})}$ (Cady)

16. Unit coal $= \dfrac{\text{B.t.u.} - 2624\text{ S}}{1.00 - (1.08\text{ ash} + 0.8\ CO_2 + 0.25\text{ S})}$ (Cady)

17. Unit coal $= \dfrac{\text{B.t.u.} - 2624\text{ S}}{1.00 - (1.08\text{ ash} + 0.4\text{ S})}$ (Cady)

18. Unit coal $= \dfrac{\text{B.t.u.} - 5000\text{ S}}{1.00 - (1.08\text{ ash} + 0.4\text{ S})}$ (Cady)

19. Unit coal $= \dfrac{\text{B.t.u.} - 5000\text{ S}}{1.00 - (1.08\text{ ash} + 0.5\text{ S})}$

20. Unit coal $= \dfrac{\text{B.t.u.} - 5000\text{ S}}{1.00 - (1.08\text{ ash} + 0.45\text{ S})}$

21. Unit coal $= \dfrac{\text{B.t.u.} - 5000\text{ S}}{1.00 - (1.1\text{ ash} + 0.4\text{ S})}$

Formulas 11 to 16 have been explained previously. Formula 17 shows modification of the determined B.t.u. value as in formula 15 and No. 16 shows modification for the pyritic sulfur only; formula 18 shows modification for the total sulfur. In formulas 19, 20 and 21 determined B.t.u. value is modified to the extent of the B.t.u. value of the total sulfur. Formula 20 was tried because it completes the series represented by formulas 18, 19 and 11.

TABLE 9.—*Comparison of Unit Coal Values Derived by Using Each of Ten Different Formulas*

Formulas	11	12	13	14	15	16	17	18	19	20	21
Range of values in units	402	466	418	468	385	426	399	400	402	392	**384**[a]
Largest number of values in any 20 units	8	5	6	4	8	6	7	7	8	7	7
Range of all but 4 extreme values	269	293	328	367	**229**	283	273	**229**	240	232	232
Number of units average value exceeds position of densest twenty	15	20	40	60	**0**	25	15	**0**	15	10	**0**
Number of values deviating more than 100 units from average value	**9**	12	13	15	**9**	10	**9**	**9**	**9**	**9**	**9**
Range of variation for individual mines											
Mine 1	**3**	70	22	86	8	55	7	11	4	5	10
Mine 2	123	139	137	125	125	145	124	127	**122**	124	123
Mine 3	63	44	95	75	45	**32**	66	44	58	51	51
Mine 4	134	168	**123**	163	146	162	130	149	138	150	139
Mine 5	**15**	81	18	34	14	31	37	45	25	25	28
Mine 6	64	162	**41**	145	76	144	61	78	69	74	83
Mine 7	52	142	74	166	42	178	53	**42**	47	44	46
Mine 8	41	35	84	88	35	**29**	44	35	36	35	37
Mine 9	46	71	**43**	53	56	51	45	57	50	53	54
Mine 10	29	15	51	**6**	24	26	29	24	27	26	29
Mine 11	23		**8**		23		10	25	16	20	28
Average range	53.9	92.7	63.3	94.1	54.0	85.3	55.1	57.9	**53.8**	55.2	57.1

[a] Lowest values in bold face type.

Comparison of the values attained by the various formulas (Table 9) indicates that there is little basis for selection among several of the formulas, particularly 11, 15, 17, 18, 19, 20 and 21. The two simplest formulas, 19 and 21, give equally good results, and results equally good and in some instances better than those obtained by the more complicated formulas. Formula 21 apparently gives slightly larger values than formula 19. In low-ash coals there would probably be no advantage of one formula over the other. The difference, however, would increase with the increase in ash so that it seems probable that for general purposes formula 19 would be more satisfactory than formula 21. Although the use of formula 19 would somewhat simplify the calculation of the unit coal value it provides no better basis for comparing coal than the original Parr formula (11), and its adoption at the present time after the Parr formula has been in use for nearly 20 years should be carefully considered from the standpoint of expediency.

It has been suggested by Dr. Fieldner that comparison is desirable between "unit coal" B.t.u. values and "pure coal" (ash-and-moisture-free) B.t.u. values for the analyses of 32 coals from Macoupin County. The "pure coal" B.t.u. values show the characteristics listed in Table 10.

TABLE 10.—*Pure Coal B.t.u. Values of Thirty-two Coals from Macoupin County*

Range of variation	369
Range of all but the four most extreme values	204
Largest number of values in any 20 units	7
Densest grouping at position of average value	
Number of values deviating more than 100 units from average value	5

Range of Variation

MINE	UNITS	MINE	UNITS
1	28	7	21
2	143	8	40
3	9	9	70
4	187	10	8
5	26	11	27
6	64	Average	56.6

The range of variation both for the entire 32 analyses and for the 28 is less than the range of variation of "unit coal" values derived by any of the 11 formulas used. The number of values deviating more than 100 units is considerably less than is the case for the values for "unit coal." These facts suggest that the "pure coal" B.t.u. values may be a better basis for comparison than "unit coal" value. However, undoubtedly the variations in values for individual mines are more significant than range of variation for a group of mines, for it is quite probable some regional variation does practically always exist, although data may not be adequate to determine its exact nature or extent. The fact, therefore, that the average mine variation for "pure coal" values is higher than for certain of the "unit coal" values is more significant than the fact that the absolute range for all analyses is less. That differences in the range of pure coal values and unit coal values for individual mines will be of the general order indicated by the data shown here, I believe has been established by numerous previous compilations as well as by the data presented in the paper under discussion.

R. P. HUDSON, Wayland, Ky. (written discussion).—The authors have written a paper of particular interest to fuel chemists and it should also be of interest to the technologist interested in the classification of coal. For purposes of classification it is desirable to know the chemical composition and calorific value of the pure coal substance.

Coal chemists are aware of the fact that the formula employed in estimating the moisture-free and ash-free calorific value falls somewhat short of perfection. A more accurate formula is desired because it would not only serve to indicate the practically exact heating value of the pure coal but would also indicate a formula for calculating the heating value of coal, which would be more accurate than any formula yet proposed. It is obvious that any formula that indicates the exact heating value of pure coal will, in its reversed form, indicate the heating value of the coal in the condition in which it was received in the laboratory.

Parr,[9] assuming his formula to be correct, states that it may be reversed to determine the calorific value of the impure coal. This can be done with sufficient accuracy to form sales contracts specifying calorific value. The formula is accurate within 100 units in 12,000, or less than a variation of 1 per cent from the values as they would be determined by direct tests in a calorimeter. The formula should be more accurate than the formula proposed by Dulong, in which the variation probably is due to the inaccuracy of the oxygen estimation required. The accuracy of Dulong's formula is approximately 1.5 per cent.[10]

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[9] S. W. Parr: Fuels Gas, Water and Lubricants, Ed. 3, 61. McGraw-Hill Book Co. New York, 1922.

The formula of Fieldner and Selvig[11] differs from Parr's original formula only in the fact that their formula considers that the sulfur in coal is not present entirely as iron pyrites. Parr's formula does not take this into consideration. Tideswell and Wheeler[12] assume that all coal contains 1 per cent organic sulfur and that the pyritic sulfur amounts, therefore, to the total sulfur minus 1 per cent. Fieldner and Selvig assume that one-half the sulfur is present in the pyritic form.

In order to obviate the necessity of assuming a value for pyritic sulfur, the writer would suggest that the sulfur present in this form be determined. The formula for non-coal then takes the following form:

$$\text{Non-coal} = M + A + \tfrac{5}{8}X + Y + 0.08\left(A - \tfrac{10}{8}X\right)$$

In this formula the factors M, A, X and Y refer, respectively, to the moisture, ash, pyritic sulfur and organic sulfur in the coal. The factor 0.08 is, as in the formula of Parr, and also in the formula of Fieldner and Selvig, a constant, representing the water of hydration for the clayey constituents.

By simplifying the formula it becomes:

$$\text{Non-coal} = M + 1.08A + 0.525X + Y$$

The "unit coal" is, therefore,

$$\text{Unit coal} = 1.00 - (M + 1.08A + 0.525X + Y)$$

We may then write the formula for unit-coal B.t.u. thus:

$$\text{Unit-coal B.t.u.} = \frac{\text{Indicated B.t.u.} - \left(\dfrac{5250}{X} + \dfrac{4050}{Y}\right)}{1.00 - (M + 1.08A + 0.525X + Y)}$$

The objection to the use of this formula is that the percentage of pyritic sulfur and organic sulfur must be known before the formula can be applied. However, these percentages may be obtained by chemical analysis. The determination of sulfate sulfur requires considerable time, but it is present in such very small amounts that it may safely be ignored. The determination of pyritic sulfur plus sulfate sulfur may be carried out in a comparatively brief period of time. The result so obtained may be considered, for our present purpose, as wholy pyritic sulfur. This result subtracted from the total sulfur in the coal will yield a figure representing the percentage of organic sulfur.

H. J. Rose, Pittsburgh, Pa.—The Parr correction diagram gives graphically the net Parr correction for ash and sulfur. This correction for coals of a reasonable degree of purity is small, since the ash and sulfur corrections are of opposite sign, and tend to cancel each other.

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[10] D. J. Demorest: Metallurgical Analysis, Ed. 5, 302. McGraw-Hill Book Co. New York, 1924.

[11] Reference of footnote 1.

[12] F. V. Tideswell and R. V. Wheeler: Pure Coal as a Basis for Classification. *Trans.* A.I.M.E. (1928) **76**, 200.

A. C. Fieldner (written discussion*).—The Sectional Committee on the Classification of Coal is indebted to Dr. Cady for an important contribution on modifications of the Parr formula for the computation of "unit" coal values. The checking of these modifications against the total of the various minerals computed from the analysis of the ash constitutes another method of appraising the relative accuracy of the formulas. While this method is not free from objections because of the necessary assumptions as to kinds of minerals present, it should serve to give an approximation of the relative accuracy of the different modifications of the Parr formula. The conclusions to be drawn from the calculated data are that a correction for carbon dioxide should be included in the formula, and that pyritic sulfur only should be included in the correction.

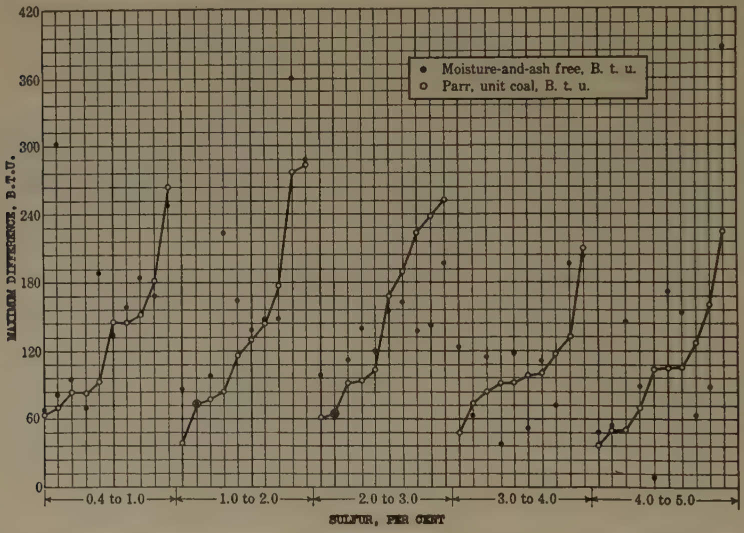

Fig. 2.—Maximum differences of pure coal B.T.U. in samples from the same mines. Five samples from each of 50 mines.

Dr. Cady next studies the merits of the various formulas for deriving a satisfactory unit-coal value by trial, assuming that the most satisfactory formula is the one that gives the least variation when applied to a group of analyses representing the same bed in a relatively small area. For this purpose, he selected 32 analyses representing No. 6 coal from 11 mines in Macoupin County, Illinois. From the calculated data given in Table 9, he concludes that the original Parr formula and several slight modifications of this formula give the most satisfactory results.

However, at the end of his discussion Dr. Cady compares moisture-and-ash-free values with the various modifications of the Parr unit-coal values as applied to these 32 analyses.

Inspection of these results shows that the moisture-and-ash-free values are practically as satisfactory from the point of view of variations in samples from the same mines as the unit values obtained with the best of the Parr formulas. The same conclusion must follow from an inspection of the computed values shown in Table 11 and plotted in Fig. 2, of 233 samples from 50 mines in various states.

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TABLE 11.—*Differences of Pure Coal B.t.u. in Face Samples from the Same Mines*

Five Samples from Each of 50 Mines in Various States

State	Rank of Coal	Percentage in Dry Coal		Maximum Difference of B.t.u. Values		Difference
		Ash Range	Sulfur Range	Moisture-and-ash-free	Parr Formula	
W. Va.	Semibituminous	3.4– 4.6	0.5–0.8	70	68	+ 2
Ill.	Bituminous	8.0–11.3	0.5–1.1	83	71	+ 12
Tenn.	Bituminous	9.0–20.9	0.5–0.7	189	93	+ 96
Utah	Bituminous	5.5– 8.0	0.7–1.0	133	145	− 12
Va.	Bituminous	6.3– 9.3	0.5–0.9	70	85	− 15
Ind.	Bituminous	6.8– 9.4	0.7–0.9	185	151	+ 34
Wyo.	Subbituminous	5.2– 6.9	0.5–0.9	248	264	− 16
N. Mex.	Bituminous	11.0–18.4	0.6–0.8	168	184	− 16
Okla.	Bituminous	4.5– 7.2	0.6–1.0	101	84	+ 17
Ala.	Bituminous	1.9– 4.1	0.7–1.0	158	146	+ 12
Ala.	Bituminous	5.3–11.5	0.9–1.3	164	116	+ 48
Pa.	Bituminous	4.5– 8.7	1.1–1.6	98	78	+ 20
Ky.	Bituminous	7.0–14.4	1.1–1.3	86	40	+ 46
W. Va.	Bituminous	6.4–11.5	1.5–2.1	74	75	− 1
Ill.	Bituminous	7.8–10.1	1.1–1.5	148	145	+ 3
Va.	Bituminous	4.0–11.1	1.0–1.4	360	278	+ 82
Utah	Bituminous	5.6– 8.5	0.9–1.4	288	283	+ 5
Ind.	Bituminous	7.4–16.1	1.3–2.1	224	85	+139
Ark.	Semibituminous	7.5–11.3	0.8–1.4	138	129	+ 9
Pa.	Bituminous	7.4–11.0	1.1–1.9	148	179	− 31
Ind.	Bituminous	8.9–12.4	2.4–2.9	66	65	+ 1
Ala.	Bituminous	5.7–11.3	1.5–3.0	163	189	− 26
Ind.	Bituminous	8.6–18.5	2.3–3.0	197	253	− 56
Ky.	Bituminous	6.8–10.6	2.4–3.4	142	238	− 96
Pa.	Bituminous	10.6–14.0	1.0–2.5	119	103	+ 16
Pa.	Bituminous	7.1–11.0	2.0–2.8	136	224	− 88
Pa.	Bituminous	9.5–12.1	1.9–3.6	99	62	+ 37
Ky.	Bituminous	8.0–14.8	2.4–3.9	140	94	+ 46
Mont.	Subbituminous	10.5–12.4	1.9–2.0	155	168	− 13
Tenn.	Bituminous	5.7–13.6	2.1–3.7	112	92	+ 20
Ill.	Bituminous	10.6–15.6	3.9–4.8	197	132	+ 65
Okla.	Bituminous	8.0–10.4	3.5–4.2	118	92	+ 26
Ind.	Bituminous	8.7–12.0	3.4–4.0	115	85	+ 30
Ky.	Bituminous	8.5–11.3	3.1–4.1	37	92	− 55
Pa.	Bituminous	12.0–16.6	3.3–4.1	63	74	− 11
Ky.	Bituminous	11.1–15.1	3.2–4.7	202	211	− 9
Ind.	Bituminous	10.1–13.6	3.3–3.8	52	99	− 47
Ala.	Bituminous	8.3–10.3	2.3–3.5	112	100	+ 12
W. Va.	Bituminous	8.3–11.7	2.7–3.9	72	119	− 47
Pa.	Bituminous	6.2– 8.8	2.2–4.1	124	48	+ 76
Okla.	Bituminous	11.0–13.8	3.7–5.1	88	160	− 72
Ill.	Bituminous	12.2–15.3	3.8–5.6	145	50	+ 95
Tenn.	Bituminous	10.9–14.0	3.6–5.0	88	70	+ 18
Ky.	Bituminous	8.7–12.3	3.5–5.9	172	105	+ 67
Wyo.	Bituminous	11.7–19.5	5.4–6.5	63	126	− 63
Wyo.	Subbituminous	14.0–19.6	5.4–5.7	153	107	+ 46
Pa.	Bituminous	8.6–11.2	5.5–6.8	55	49	+ 6
Pa.	Bituminous	10.9–19.8	4.2–5.6	386	223	+163
Tenn.	Bituminous	6.9–10.1	3.6–4.4	48	36	+ 12
Ind.	Bituminous	10.4–13.3	4.4–6.1	9	103	− 94
	Arithmetical mean =			135	125	41
	Algebraic mean =					+ 10
	Maximum difference			386	283	163

	Moisture-and-ash-free	Parr
Percentage of mines differing less than 200 B.t.u.	88	84
less than 250 B.t.u.	94	92
less than 300 B.t.u.	96	100

TABLE 12.—*Differences in Pure Coal B.t.u. in Face Samples of Anthracite from the Same Mines*

Three to Six Samples from the Same Mine.

Mine No.	Maximum Difference of B.t.u. in Pure Coal		Difference between Moisture-and-ash-free and Parr Formula
	Moisture-and-ash-free	Parr Formula	
1	20	50	− 30
2	20	20	0
3	80	40	40
4	130	120	10
5	40	0	40
6	170	30	140
7	20	110	− 90
8	60	60	0
Average	68	56	
Maximum	170	120	

The average of the maximum differences of moisture-and-ash-free calorific values for the 50 mines was 135 B.t.u. as compared to 125 B.t.u. by the Parr formula; 88 per cent of the mines showed differences of less than 200 B.t.u. on a moisture-and-ash-free basis as compared to 84 per cent on the Parr unit-coal basis.

Table 12 gives similar data for eight anthracite mines. For these mines the average of the maximum differences, moisture-and-ash-free, is only 12 more than the average for the differences by the Parr formula.

All of these statistical studies show that, in spite of the known absolute errors in pure-coal values computed on the basis of "uncorrected" ash, the ash-free values of different samples from the same mine agree almost as well on the ash-free basis as on the Parr unit-coal basis. How can we explain this apparent anomaly? It is known that the type and composition of the different layers in a coal bed may vary and that these layers vary in relative thickness in different parts of a mine. Consequently, the mean composition of the coal substance in a composite face sample of coal varies at different points in a mine over a range that is of the same order or larger than the errors introduced by the use of the more approximate ash-free formula.

The magnitude of this variation has an important bearing on the accuracy required in the determination of the pure-coal analytical values used in the classification of coal. From a practical point of view it may be needless to use complicated correction formulas or analytical processes if the normal range of variation in composition of the coal substance in a given mine is of greater magnitude than the admitted error of the ash-free values.

In this connection, it is interesting to know the theoretical differences in ash-free values caused by certain ranges of variation in ash and sulfur in coal on the assumption that the Parr formula gives the true pure-coal values. These theoretical differences may be calculated as follows for two coals of the same pure-coal composition:

	COAL A	COAL B
Unit B.t.u.	15,000	15,000
Unit fixed carbon, per cent	80.0	80.0
Ash, dry coal, per cent	2.0	10.0
Sulfur, dry coal, per cent	0.5	2.5

$$\text{Unit B.t.u.} = \frac{\text{Dry B.t.u.} - 5000\ S}{1 - (1.08\ \text{ash} + 0.55S)} \qquad [1]$$

Substituting unit values for coals A and B and solving,
Dry B.t.u., coal A = 14,660

$$\text{Moisture-and-ash-free B.t.u. coal A} = \frac{14{,}660}{1.00 - 0.02} = 14{,}959$$

Dry B.t.u. coal B = 13,299

$$\text{Moisture-and-ash-free B.t.u. coal B} = \frac{13{,}299}{1.00 - 0.10} = 14{,}777$$

Difference, moisture-and-ash-free B.t.u. coals A and B = 14,959 − 14,777 = 182

$$\text{Unit fixed carbon} = \frac{\text{Dry fixed carbon} - 0.6\ S}{1.00 - (1.08\ \text{ash} + 0.55\ S)}$$

Substituting unit values for coals A and B, and solving,
Dry fixed carbon, coal A = 78.4

$$\text{Moisture-and-ash-free fixed carbon, coal A} = \frac{78.4}{1.00 - 0.02} = 80.0$$

Dry fixed carbon, coal B = 71.8

$$\text{Moisture-and-ash-free fixed carbon, coal B} = \frac{71.8}{1.00 - 0.10} = 79.7$$

Difference ash-free fixed carbon, coals A and B = 0.3

In a similar manner, differences have been computed for smaller ranges of sulfur and ash. The results are given in Table 13. This table shows that if samples for coal classification are confined to those containing less than 10 per cent ash and 2.5 per cent sulfur, the maximum variations possible in moisture-and-ash-free values as compared to unit-coal values are 182 B.t.u. and 0.6 per cent fixed carbon.

TABLE 13.—*Theoretical Differences in Moisture-and-ash-free Values*

Range of Analysis			Maximum Difference	
Case No.	Percentage of		B.t.u.	Fixed Carbon
	Ash	Sulfur		
1	2 to 10	0.5 to 2.5	182	0.3
2	2 to 10	0.5 to 2.0	165	0.4
3	2 to 8	0.5 to 2.0	134	0.2
4	2 to 8	0.5 to 1.5	115	0.3
5	2 to 6	0.5 to 1.5	87	0.2
6	2 to 6	0.5 to 1.0	70	0.2
7	2 to 10	0.5 to 0.5	110	0.6

If coals contain carbonates, or more than 10 per cent ash or 2.5 per cent sulfur, they should be floated on a heavy solution to bring the ash down to about 6 or 7 per cent.

On the basis of the foregoing discussion it might be advisable to recommend that the proximate and ultimate analyses and the heat of combustion when used in the classification of coal should be referred to the ash-free basis, and that samples suspected of containing carbonates, or those containing more than 10 per cent ash or 2.5 per cent sulfur in the dry coal, should be crushed and floated on a heavy liquid of such specific gravity as to reduce the ash in the float coal to less than 10 per cent—provided, however, that the float coal recovery should not be less than 75 per cent.

The classification of coal from a given mine and bed should be based on the average ash-free analysis of not less than three and preferably five face samples taken from the same bed in different parts of the mine.

If these recommendations were adopted, and if boundary lines of fixed carbon and B.t.u. were set up between adjacent classes of coal it would then be advisable to permit a tolerance of ±200 B.t.u. and ±1 per cent fixed carbon at the boundary. Coals falling between these limits, other factors being equal, could be classified on either side of the boundary. In effect, this plan puts a band 400 B.t.u., or 2 per cent fixed carbon, wide between adjacent classes of coal. Coals falling on the band may be in either class. These tolerances are ample to cover errors of sampling and analysis, and the deviation of ash-free from pure-coal values in the range of ash and sulfur specified.

Properties of Coal Which Affect Its Use in the Ceramic Industry*

By W. E. Rice,† Pittsburgh, Pa.

(New York Meeting, February, 1932)

The ceramic industry has to do with forming or molding articles of clay, and imparting to them their characteristic properties of permanence, strength and color by subjecting them to heat treatment in kilns. This heat treatment is commonly known as "firing" or "burning." The art of firing clay ware antedates the industrial use of coal; hence wood was the fuel first used, and the earliest kilns were built to burn wood. The majority of the clay ware produced today is fired in periodic kilns that are essentially enlarged structures similar to the earliest enclosed kilns; a kiln consists of a large chamber 16 to 32 ft. dia. (or rectangular) in which the ware is charged and fired either in direct contact with flame or hot gases from the fuel, or in saggers which shield the ware from direct contact with the gases. The fuel is burned in a number of small furnaces built in the thick wall that encloses the chamber; these furnaces are from 16 to 36 in. wide and from 3 to 5 ft. long; some of them have horizontal or sloping grates, but many have no grates; the last are called "dead-bottom" furnaces. The temperature necessary to mature the charge in the kiln varies for different clays; for some it is as low as 1800°, and for others as high as 2700° F. Wood was an ideal fuel for the low temperatures that were used in early kilns; it burns freely and produces a long, clean flame, which promotes uniform distribution of the heat in the kiln; it does not coke and so restrict the passage of air through the fuel bed; the ash does not melt in the fuel bed to form clinker; and the ash that is carried into the chamber and deposited on the charge does not contain iron and sulfur to discolor the clay.

The extent of the ceramic industry now precludes the general use of wood in kilns, so that today this industry is one of the large users of bituminous coal, burning more than eight million tons annually in kilns. Manufacturing plants usually are at or near the source of clay, with deposits of coal suitable for kilns occurring near the clay. The cost of coal is the largest single item of expense in the manufacture of clay ware, and, because all coals are not equally suitable for the process, the careful selection of coal is important. In this selection the desideratum is to produce the highest possible proportion of high-quality ware at the lowest possible cost. The cost of freight is an important factor, but for the present discussion can be considered as included in the cost of the coal. Other than the cost, the important factors that affect the selection of coal for kilns are (1) the effect of the properties of the coal on the quality of the fired clay, (2) the effect of the properties of the coal on the labor required to fire the kiln, and (3) the effect of the properties of the coal on the rate at which the kiln can be heated.

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Qualities of Coal for Periodic Kilns

The qualities that render a coal suitable and desirable for use in periodic kilns, and in other kilns in which the coal is burned in small hand-fired furnaces, are as follows:

Volatile Matter.—The process requires uniform heat treatment to mature every part of the charge in the kiln. This can be accomplished most easily and economically with a coal which has a high content of volatile matter and a high calorific value, and which burns with a long radiant flame, so that a large area of the charge is heated by radiation. With such a coal, burned in comparatively deep fuel beds, the firing of a kiln is done without overheating the furnaces, which can then be used for long periods without repair. When, however, a low-volatile coal or a coke is used it is necessary to have wider furnaces; *i. e.*, a greater ratio of grate area to kiln area and a correspondingly slower rate of burning, and deeper fuel beds; yet the firing is not as satisfactory because of overheating of the areas near the furnaces.

During the first few hours of the firing, the kiln and charge are cool and moist; when a long flame meets such a surface, combustion stops and soot is deposited. This happens in spite of the fact that from 500 to 1500 per cent excess air passes through the furnaces. The deposit of soot, although it later burns, sometimes affects the color of the matured product; sometimes the deposit of soot becomes so dense that it clogs the passages between the piled ware through which the gases flow, and stops the operation of the kiln. Experience indicates that sooting is more likely to occur with bituminous coals when the fuel ratio (fixed carbon divided by volatile matter) of the coal is lower than 1.5 than when it is higher. In a few kilns anthracite, coke, or low-volatile coal is used until the kiln is hot (1000° F.) and thereafter high-volatile coal is burned, but this practice is unusual; it is more necessary when the production of smoke is considered a nuisance.

The color of clay ware is affected by the gases to which it is exposed during firing, and particularly during the final stage of firing, when it is at the maturing temperature; this is owing to the reduction or oxidation of oxides of iron in the clay by the gas. Usually the clear colors produced by oxidation are desired; hence the volatile matter as it affects the length of flame is limited because, even though it is desirable to have as long a flame as possible to radiate to the charge, combustion should be completed before the gas strikes the charge so that the iron will not be reduced. In some instances, however, reduction is desired and coals with a higher content of volatile matter are used successfully.

Sulfur.—During firing clay is sometimes discolored by white deposits, called "scum," which are formed by the combination of lime in the clay with sulfuric acid from the sulfur in the coal. This does not occur if the clay does not contain lime, or if the concentration of sulfur in the gas in the kiln is low; hence coal that contains a low percentage of sulfur is desirable. Further, coal that contains a high percentage of sulfur causes the concentration of sulfur dioxide in the air around the furnaces to be so high that the firemen object to the use of the coal or do not keep the fires clean.

Ash and Moisture.—A low percentage of ash in the coal burned in kilns is desirable; for, if the content of ash is high, the furnaces must be cleaned frequently, which delays the progress of the heating of the charge, and, because it necessitates some agitation of the fuel bed, causes a greater quantity of ash to be carried from the furnace with the gas and deposited on the charge, where it may cause discoloration or otherwise damage the surface of the clay. This damage is most likely to happen if the ash is red, for the red color is caused by the iron in it; it is not likely to happen if the ash is white and flaky like wood ash, for, although more of the latter ash may be carried from the furnace, it contains little iron and sulfur, which are the constituents most likely to damage clay.

The moisture is a less important consideration than the ash; but it lowers the calorific value of the coal and renders part of the heat developed unavailable for heating the kiln, by converting it to latent heat; therefore a low percentage of moisture is desirable.

Caking and Coking Qualities.—A coal that does not form a compact mass quickly after it is placed in a hot furnace or does not form solid masses of coke is desirable, for such qualities reduce the flow of air through the fuel bed and furnace, making it difficult to establish and maintain the desired rate of burning and the correct atmosphere in the kiln.

Physical Qualities.—The softening temperature of the ash should preferably be 2600° F. or higher, although an ash of which the softening temperature is between 2300° and 2600° F. is fairly satisfactory if the percentage of ash in the coal is not high; in most of the coals used the softening temperature of the ash is within the lower range. Ash of low softening temperature forms clinker easily, and thereby reduces the flow of air through the fuel bed and makes it necessary to clean the furnaces frequently, especially if the coal contains much ash; thus the labor required is increased and the ash deposited on the charge is more likely to damage the ware because of its lower fusibility.

Sized coal is preferred, and the 2 by 4-in. egg is the size that generally meets the requirements; the 1½ by 3 and 3 by 6-in. sizes have also given good results. The coal must be hard enough to withstand the handling between the tipple and the kiln if the advantages of buying a prepared size are to be realized. At the beginning of the cycle of firing the kiln and stack are cool; hence only a small draft is available to provide the air to burn the coal; at this time the larger sizes are obviously preferable, for they allow the low draft to draw more air through the fuel bed so that the heating can be done as rapidly as the condition of the ware permits. The large sizes are also less likely to produce much smoke at this stage, when sooting may occur. The use of screened coal gives a fuel bed that burns more uniformly over its area, and also between times of firing; in addition, the admittance of air through and over the main fuel bed is regulated in most kilns by properly locating the lumps of coal.

Summary of Qualifications

Based on the considerations that have been stated, an ideal coal for the ceramic industry has the proximate analysis and qualities shown in Table 1. The fuel ratio of this coal is 1.56; it should not be less than 1.5. The coal should be hard enough to withstand handling.

TABLE 1.—*Characteristics of Ideal Coal for Ceramic Industry*

	Per Cent		
Moisture	2	Sulfur	not more than 1 per cent
Volatile matter	36	Calorific value	14,000 B.t.u. per lb.
Fixed carbon	56	Softening temperature of ash	2600° F.
Ash	6	Color of ash	white
	——	Size	2 × 4 in.
	100		

These properties are closely the same as the requirements for a good gas coal described by Francklyn,[1] with the exception that the property of strong coking is not desirable in coal for kilns.

DEVIATIONS FROM THE IDEAL QUALIFICATIONS

Many kilns are fired with coal that differs widely in one or several properties from those which have been named as desirable. Often first cost of coal is the only factor that is seriously considered; the manufacturer is fortunate if he obtains satisfactory coal by this method of buying. Some products are low priced and are salable at the same price regardless of the differences in color and hardness that result from the firing; *e. g.*, common building brick. For these the selection of coal is not governed as much by the properties as by the price. Some clays develop beautiful colors when sooted, or flashed by firing in a reducing atmosphere; and for these coals having a lower fuel ratio can be used. Many products are not seriously affected by ash deposited on them in the kiln, so that the coal used to fire them may be below the requirements stated for the quantity and softening temperature of the ash.

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[1] G. Francklyn: Properties of Coal Which Affect Its Use for the Manufacture of Coal Gas, Water Gas and Producer Gas. *Trans.* A. I. M. E., Coal Div. (1930) 706–712.

The properties of six coals for kilns are given in Table 2.

TABLE 2.—*Properties of Coals for Kilns*

	Coal Number[a]					
	1	2	3	4	5	6
Proximate analysis:						
Moisture, per cent	2	3.6	7.5	4.4	3.3	7.2
Volatile matter, per cent	36	35.6	32.5	40.9	38.7	31.6
Fixed carbon, per cent	56	54.6	57.0	45.4	50.9	50.2
Ash, per cent	6	6.3	3.0	9.3	7.1	11.0
Sulfur, per cent	1	1.0	0.8	3.1	2.5	1.0
Fuel ratio, fixed carbon/vol. matter	1.56	1.53	1.75	1.11	1.32	1.59
Calorific value, B.t.u. per lb	14,000	13,540	12,500	12,510	13,430	11,790
Softening temperature of ash, deg. F	2,600	2,460		2,730	2,430	2,190

[a] No. 1 is the ideal that has been given in this paper.

No. 2 is the average of 80 coals of the United States that are considered good coals for kilns.

No. 3 is representative of a coal of the Appalachian region of the eastern province that is said to be one of the best in the United States for kilns. This coal has not the hardness demanded of the ideal coal, but it does not coke in the furnace, and the softening temperature of the ash is high.

No. 4 and 5 are coals that are used to fire bricks made of clay that is not damaged by sooting, reducing atmosphere or sulfur. The fuel ratios of both are low and they produce dense smoke in the kiln. No. 5 cokes quickly in the furnace, hence is more satisfactory in wide than in narrow furnaces.

No. 6 is a coal from the eastern region of the interior province. This gives satisfactory results in kilns of ware that is not affected by the ash. It does not coke in the furnace. Because it forms clinkers easily, it can be burned in wide dead-bottom furnaces better than in narrow furnaces or on grates.

PROPERTIES OF COALS FOR USE IN OTHER TYPES OF KILNS

In some continuous kilns the coal is burned in the chamber with the ware; the use of these kilns is limited to low-quality products, and the properties of the coal that can be used are not so limited. Continuous and regenerative kilns with hand-fired furnaces use coal with properties as stated in the foregoing. Some kilns for common building brick and other low-grade products burn low-grade slack, or slack and nut coal in small forced-draft furnaces. These may blow much ash into the chamber.

A few kilns have been built to be fired with stokers and a few to be fired with pulverized coal. These have not been satisfactory because of the ash factor; some because of the quantity of ash deposited on the charge, and others because the softening temperature of the ash was so low that trouble owing to clinker was excessive.

Producer gas is used in a number of continuous and regenerative kilns, and also in a few periodic kilns. The properties of coal for the manufacture of producer gas as given by Francklyn are applicable when the gas is to be burned in kilns. When the gas is burned in regenerative kilns of the continuous chamber type emphasis should be placed on the requirement for coal low in sulfur; for, if the sulfur is high, the operation is such that the gas leaving the kiln contains a high concentration of sulfuric acid which damages the fans used to move the exhaust gas, the flues, and stack. High sulfur is harmful to vegetation in the vicinity of any type of kiln.

DISCUSSION

(*H. J. Rose presiding*)

R. S. Julsrud, Philadelphia, Pa. (written discussion).—Based on the author's specifications of ideal coal for the ceramic industry it appears that anthracite possesses many merits, as shown in Table 3.

Table 3.—*Comparison of Anthracite with Ideal Coal*

	Ideal Coal	Anthracite, Steam Sizes
Moisture, per cent	2	1.5 to 3.5
Volatile matter, per cent	36	3.0 to 10
Fixed carbon, per cent	56	80.0 to 90
Ash, per cent	6	8 to 15
Sulfur, per cent	1	0.5 to 1
B.t.u. dry	14,000	13,500 to 14,000
Fusing temperature of ash, deg. F	2,600	2,600 to 3,000
Color of ash	White	White and red
Size, in	2 × 4	½ and down

Many kiln operators in Pennsylvania, New York, and New Jersey formerly used the domestic size of anthracite and obtained excellent kiln operation and quality of product. As the domestic heating field claimed these sizes and the use of smaller or steam sizes with forced draft was not generally understood, the kiln operators turned to bituminous.

The present excellent preparation of steam-size anthracite commends this fuel either hand-fired, stoker-fired or in a gas producer, according to the plant requirements. Among its merits may be listed: (1) low hydrogen content, offering very dry combustion products; (2) freedom from smoke; (3) low sulfur content; (4) competitive prices with other fuels in certain districts; and (5) high radiant heat value of anthracite flames.

The long-flame feature of high-volatile fuels can be secured with suitable furnace design to produce the same net "reach" results when burning anthracite. One manufacturer is securing excellent results burning anthracite, with coal consumption comparing favorably with bituminous plants.

Another industry in which the particular fuel specifications conform very closely to the properties of anthracite is the manufacture of solid CO_2, or "dry ice." Here a fuel low in sulfur, hydrogen and volatile is required in order that the products of combustion, from which the CO_2 is made, may be recovered with a minimum of scrubbing and washing.

H. G. Turner, State College, Pa. (written discussion).—Without having at my disposal several types of ceramic plants for experimental work, using many kinds of coal, I would be very reluctant to draw up specifications for an ideal coal for the ceramic industry. It seems to me that the ideal coal would be the one that turned out the best finished product at a cost lower than could be produced by any other coal. If that is so, Pennsylvania anthracite is the ideal coal in Hazleton, Pa., where it is being used with marked economy over other fuels in a modern plant engaged in the manufacture of common and face brick of high quality. As a matter of fact, a recent study made by the Anthracite Institute shows that the small sizes of Pennsylvania anthracite, either hand-fired with forced draft or used in producers, is the ideal fuel for the ceramic industries located in the normal anthracite sales area. Anthracite has all the ideal qualifications listed by Mr. Rice, with the exception of high volatile matter. The desirable features to be gained by the high volatile content can easily be brought about through proper design of the anthracite furnace.

H. E. Nold, Columbus, Ohio (written discussion).—Mr. Rice's paper contains many things of value and should be read by all executives of ceramic plants who are responsible for the proper and economic firing of clays in periodic kilns. It seems to the writer, however, that certain of the assumptions and conclusions in this paper might be challenged. In the first paragraph the following statement is made, "Wood was an ideal fuel for the low temperatures that were used in early kilns; it burns freely and produces a long, clean flame, which promotes uniform distribution of the heat in the kiln." Also, regarding volatile matter, the author says: "The process requires uniform heat treatment to mature every part of the charge in the kiln. This can be accomplished most easily and economically with a coal which has a high content of volatile matter and a high calorific value, and which burns with a long radiant flame, so that a large area of the charge is heated by radiation." It is difficult to understand how the long wood flame promotes uniform distribution of the heat in the kilns. It is well recognized, in the ceramic industry, that heat distribution in the kiln depends upon sufficient and properly controlled draft and upon the proper setting of the ware in the kiln so that the openings between the ware, through which the heated gases must pass, are so located and so proportioned that the flow of the heated gases will be uniform throughout all of the ware in the kiln, thus assuring equal heat to all parts of the kiln. Mr. Rice's statement that the long coal flame heats a large area of the charge by radiation would seem to need experimental demonstration. The writer is well aware that many kiln firers believe a long flame necessary in order that the combustion of the gases shall be completed in the narrow passages between the pieces of the charge. It seems probable that by far the greater amount of the heat is carried to the setting by convection. The luminosity of the gases in the openings between the materials set in the kiln, caused by particles of carbon heated to incandescence, is an evidence of incomplete combustion and accounts for smoke and wasted energy in the coal. May it not be true that the heat so lost by incomplete combustion may more than offset any heating by radiation from the flame?

On page 250 Mr. Rice says that at the beginning of the cycle of firing the larger sizes of coal are preferable. This statement is true when applied to many of the smaller and some of the larger plants. In modern plants, however, the practice is to connect a number of kilns to one stack. Thus, by proper rotation of the firing of the kilns, the stack is always hot and ample draft is at all times available, making unnecessary the use of the larger and more expensive fuel. In many of the older plants where the stack connection mentioned above is not available the necessity for using the coarse and more expensive coal during the first part of the burn is eliminated by using induced draft.

J. T. Robson, Columbus, Ohio (written discussion).—In paragraph one, the author says that the majority of clay ware produced today is fired in periodic kilns. This should be made to read "The majority of *heavy* clay wares." The heavy clay group consists of face brick, common brick, paving brick, refractories (sometimes), hollow tile, sewer pipe, drain tile, flower pots and roofing tile. While it may be true that the majority of all clay wares are fired in periodic kilns, this is most certainly not true in some other ceramic fields such as semi-porcelain, artware, possibly vitrified china and others, where tunnel kilns have largely replaced periodic kilns. The author mentions also the use of saggers. This automatically includes the ceramic fields mentioned above, since saggers are not used in the heavy clay field.

In the second paragraph, he says, "The cost of coal is the largest single item of expense in the manufacture of clay ware, etc." Actually the labor cost is the largest single item.[2]

On page 248, the statement is made, speaking of low-volatile coal, " . . . yet the firing is not as satisfactory because of overheating of the areas near the furnaces." Very often where a straight nonflashed oxidized product is desired, a low-volatile coal is and must be used for firing heavy clay products; also, in the case of forced-draft kilns where exceptionally uniform results are obtained in the kiln and in the firing of glazed ware, a low-volatile coal is desirable and exceptionally uniform firing is obtained. We think the above should read, " . . . yet the firing *sometimes* is not as satisfactory, etc."

On page 249, the statement, "During firing, clay is sometimes discolored by white deposits called 'scum' which are formed by the combination of lime in the clay with sulfuric acid from sulfur in the coal," should read, " . . . deposits called scum which *often* are formed," since very often this scum is due to magnesia reacting with the sulfur gases from the coal or sometimes from pyrite.

On page 252 the author says: "A few kilns have been built to be fired with stokers and a few to be fired with pulverized coal. These have not been satisfactory because of the ash factor, etc." We have several stoker-fired kilns, one handling one-fired glazed porcelain, where the results are entirely satisfactory, and also forced-draft installations that are highly satisfactory, which, in the same paragraph, the author says "may blow much ash into the chamber."

These criticisms are offered in a friendly manner. The analyses and characteristics of desirable coals for ceramic work make a very valuable addition to ceramic literature.

G. H. Cady, Urbana, Ill.—Certain mid-western coals contain sodium chloride in sufficient amount to produce an undesirable glaze on some ceramic ware, according to Parmelee and Hurst, as reported at the Mid-West Bituminous Coal Conference at Purdue in 1930.

R. H. Sweetser, New York, N. Y.—Mr. Rice's specifications for coal in the ceramic industry almost hold for coal for blast-furnace coke and for good gas coke, and apparently other industries want low-ash, low-sulfur, in other words, *clean* coal. The arguments advanced here for good coal in the nonferrous industries hold equally well among blast-furnace men. All want the best quality at the lowest cost. However, the specifications are hard to meet without mechanical cleaning, and much of the coal used in the Ohio ceramic industry fails to meet them. Furthermore, uniformity is vital to satisfactory operation, particularly in blast-furnace work.

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[2] See *Brick & Clay Rec.* (July 3, 1928).

W. E. Rice (written discussion).—The properties given for an ideal coal define a coal that will give best results in kilns under ordinary conditions and with least care. It is not to be assumed that all ceramic industries are situated so that they can economically use a coal which has all these properties, hence the real problem is to develop furnaces and methods of firing and control so that they can most closely approach the desired results with other coals, which are desirable because of cost (in which availability is an important factor) or other reasons. Comparison of the properties of an ideal coal with those of another coal, which differs in one or several properties, will indicate the nature of the precautions that should be taken in construction of furnaces and method of operation in order to obtain suitable results with the latter coal.

Many data have been published on radiation from flames, some based on experimental studies to determine flame temperatures and some based on results in furnaces; these show that at high temperatures (above 1000° F.) much heat is transferred by radiation, and this should hold in kilns. The carrying of part of the heat of the coal away from the furnaces as undeveloped heat in combustible gases and the liberation of it by combustion beyond the bag walls means that the furnaces are cooler and the heat is made available closer to the place where it is used. When the temperature of a kiln is high the loss owing to sensible heat in the flue gas is correspondingly high; with reasonable care to admit enough air to complete combustion in the kiln, the loss owing to combustible in the flue gas should not be as great as would be the increased loss in sensible heat if convection alone were depended upon for transfer of heat from furnaces to charge.

The paper cited by Mr. Robson shows that the total cost of labor after the clay is delivered to the plant is greater than the cost of coal. Confidential data that we have show the following order of costs: Yard overhead (highest), burning, brickmaking, loading, setting, drying (lowest). The statement made in the paper is based on a system of cost accounting in which the various steps in the manufacture are fully subdivided, the overhead excluded because it is not a direct expenditure, and the loading excluded because it is subsequent to the completion of manufacture.

Use Classification of Coal in the Portland Cement Industry

By H. P. Reid,* Chicago, Ill.

(New York Meeting, February, 1932)

Portland cement is manufactured under either of two general processes, the wet or the dry. The raw materials in general consist of limestone, shells, marl, cement rock, clay, shale, blast-furnace slag, etc., so proportioned that when calcined the desired composition will be obtained.

In the wet process the combined raw materials are ground to extreme fineness in the presence of water, to a resulting slurry of from 30 to 50 per cent moisture content. The moisture of the slurry may be reduced by thickening or filtering processes for feed to the cement kiln or the slurry may be fed to the kiln with its full percentage of water.

The dry process differs from the wet in that the surface moisture of the raw materials is removed prior to grinding. For the drying of these materials heat is required and in most cases is produced by burning of raw fuel either on grates or stokers before the dryers or in suspension in the form of pulverized coal, oil or gaseous fuel.

The power required for making the cement is sometimes purchased from outside sources; sometimes it is generated from the heat in the waste gases leaving the cement kilns, or generated at the plants direct from raw fuel.

We have, then, as the possible requirements of fuel in the cement industry: (1) fuel for power generation, (2) fuel for drying raw materials, including coal before pulverizing, and (3) fuel for burning the raw materials to cement clinker. There is no essential difference in quality required in fuel for burning the materials by the wet and dry processes.

Fuel for Power Generation

As there is nothing peculiar to steam generation in a cement mill power plant as compared with other power plants this use classification of such fuel will be found in other reports of the Technical Committee.

Fuel for Drying Raw Materials and Coal

Any fuels capable of producing moderate temperature gases may be used for drying raw materials. There is no problem of material contamination from fuels at this part of the cement manufacture to be guarded against.

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* Special Engineer, Universal Atlas Cement Co.

With proper furnaces any coals may be burned by hand firing or on stokers to supply necessary heat for removal of moisture from raw materials. When the moisture content of raw materials is high it usually is necessary to dilute the products of combustion with air so as to provide the necessary moisture-carrying vehicle. For drying of coal to be pulverized, the combustion gases must be tempered before they contact with the raw coal to prevent either the distillation of some of the volatile matter of the coal or the actual ignition of the coal in the direct type of dryer.

When pulverized coal is used for drying raw materials, combustion space should be provided to prevent the quenching of the burning coal particles by the rock dust in the dryer. In some cases the coal is burned within a pulverized coal furnace and the products of combustion diluted with air between the furnace and the dryer, while in other cases the fuel is burned in suspension within the dryer. In the latter, the temperature of the combustion space is not sufficient to hold the flame, making refractory-lined ignition chambers necessary. Generally speaking, combustion is poor in the latter case and combustible losses are liable to run high.

Fuel for Burning Raw Materials to Cement Clinker

Practically all portland cement clinker made in the Americas is burned in a rotary kiln, which is a large nearly horizontal cylinder of steel lined with refractory. The kiln is supported by tires on rolls and is turned about the main axis of the cylinder. It is set with a slight pitch, varying from ¼ to ¾ in. per foot of length, which causes the material to travel along the length of the cylinder as the kiln is rotated.

The raw feed, wet or dry, is fed into the high end of the kiln and fuel is burned in air suspension at the lower or material discharge end of the kiln. In wet process kilns, the moisture in the wet slurry entering the kiln is evaporated by the gases which have been partly cooled in passing over the material in the kiln below this point. After the moisture in the raw feed or slurry has been evaporated the action of the heat on the material is similar for either wet or dry process.

The material in its travel down the kiln is preheated until its temperature has been raised sufficiently to drive off the carbon dioxide from the calcium and magnesuim carbonates and to burn out any organic compounds. The temperature of calcination varies with the CO_2 partial pressure of the gas; however, it is generally conceded that at 1700° to 1750° F. most of the CO_2 has been removed. The material is then further heated to effect desired chemical combinations of the calcium, silica, alumina, etc., compounds. These combinations are exothermic but the exact value of the heat evolved varies with the composition of the feed and possibly also with the form in which the various constituents exist in the raw materials. To complete the desired chemical formation the material is further heated to from 2500° to 2700° F.

From the above, it is evident that whatever fuel is used for burning portland cement clinker that fuel must be capable of producing flame temperatures sufficiently high to raise the material to 2700° F. Next, the products of combustion after they pass the hot zone must have sufficient heat content to do the work of calcining the carbonates at material temperatures of 1700° F. or greater. Inasmuch as time element enters into the formation of the desired chemical changes, the flame should be long and luminous for the most economical production of clinker. Where the heat can be maintained over a long burning zone with a given composition of material the maximum temperature required may be considerably lower than if the flame has a short intense zone. Furthermore, the short intense zone is liable to punish the refractory.

Because the pulverized coal is burned in suspension in a horizontal blast and also because it is not practicable to obtain much gas turbulence inside the kiln, it is desirable to grind the coal much finer than is the common practice for power-house purposes. For best results coal as fired to a cement kiln should contain a very low percentage of particles over 100-mesh size and 80 per cent or more should pass the 200-mesh screen.

The effect of sulfur in the coal varies according to the nature of raw materials used. With some raw materials apparently coals having a sulfur content up to 5 per cent or greater may be used, while with other raw materials such a sulfur content might not be acceptable. The sulfate content of standard portland cement is limited by specifications to 2 per cent. Sulfate in the form of gypsum must be added to the clinker as it is ground to cement in order that a control may be maintained over the setting time of that cement. If the clinker has absorbed sulfur from other sources, the amount of setting control may be limited thereby.

In general the ash content of the coal, as long as it remains nearly constant, is not detrimental to clinker burning except as the ash reduces the flame temperatures, or affects the burning characteristics of the coal. However, the chemist must always take the ash into consideration in making his composition control. Since the ash enters into the clinker it should be considered when purchasing coal, as it may prove expensive raw material for cement manufacture.

While various types of coal burners have been tried in the cement industry, the open round pipe is used almost exclusively. Primary air, varying from 20 per cent up to nearly 50 per cent, is introduced with the coal blast. The percentage varies with the characteristics of the coal, the draft available on the kiln, the temperature of the combustion air, etc.

Sufficient heat remains in the clinker as it leaves the kiln to preheat the combustion air from 700° to 1200° F., according to the fuel practice and the design of the kiln. The lower the fuel practice, the higher will be the obtainable preheat. Clinker coolers or heat exchanges have been installed with most of the modern equipment and are being added to many of the older installations. Yet there are many of these older installations where space is not available for such added equipment.

The fuel practice varies widely throughout the industry. Plants that are able to produce portland cement clinker at a fuel rate below 1,000,000 B.t.u. per barrel of 376 lb. are the exception, while with some raw materials wet processed a fuel rate of 2,000,000 B.t.u. per barrel may be necessary.

DISCUSSION

(*H. J. Rose presiding*)

R. S. Julsrud, Philadelphia, Pa. (written discussion).—Adjacent to the large Lehigh cement district of Pennsylvania lie large deposits of anthracite silt and culm. The use of this fuel fired either in pulverized form or in gas producers has proved highly satisfactory to many industries. A large public utility in Pennsylvania is at present firing pulverized river anthracite with pulverizing costs of approximately 24¢ per ton and maintenance costs of less than 2¢ per ton. Favored with short freight haulage and low costs at the mines, it appears that cement operators in this district might profitably consider this fuel in connection with their power and kiln requirements.

H. P. Reid (written discussion).—Pulverized anthracite has been tried a number of times for burning cement in rotary kilns but thus far has not been successful except when mixed with a high-volatile coal. The low volatile content of anthracite makes it very difficult to ignite and hard to hold the flame in a rotary kiln. Generally speaking, the tendency with such a fuel is to throw the fire back into the kiln and thus shorten the effective length of that machine. We would be inclined to feel that producer gas made from anthracite should be a good cement-burning fuel if it could be produced in competition to a pulverized bituminous coal. There is no doubt that a careful study of the application of anthracite in pulverized form, or as producer gas, to the burning of cement would develop some methods of firing that would overcome these difficulties and allow at least a high percentage of anthracite to be used if not a straight anthracite fuel.

Qualities of Coal and Coke Required in Nonferrous Metallurgical Industries

By Clyde E. Williams,* Columbus, Ohio

(New York Meeting, February, 1932)

The purpose of this report is to supply the Coal Classification Committee[1] with information for use in the formulation of a system of classification of coals according to their uses. Consideration is given here only to the nonferrous metallurgical operations that consume relatively large amounts of coal or coke; *i. e.*, the smelting of copper, lead, zinc and aluminum.

For all of these uses, coal of certain definite qualities is desired. Geographical location, however, often determines whether these qualities can be specified. In some instances the preferred qualities must be specified, regardless of cost. In others, quality may be sacrificed for lower cost, although the best quality consistent with cost is usually the determining factor. These considerations will be outlined in the following discussion under the various uses, which, however, will illustrate the impracticability of preparing definite specifications for coal used in the majority of the nonferrous industries.

Coal and Coke in Copper Metallurgy

From the standpoint of coal consumption, copper smelting and refining operations are the most important of the nonferrous metallurgical uses of coal. As nickel ores treated in the United States and Canada are associated with copper, the first step in their recovery is really a copper-smelting operation; accordingly, the following considerations under copper smelting apply equally to nickel.

Copper Smelting in Reverberatory Furnaces.—For a long time, reverberatory copper-smelting furnaces were fired with lump coal burned on grates in a fire box. Later, lump coal was displaced by pulverized coal or oil burned directly in the smelting chamber. Recently, natural gas has been used in several plants, although producer gas was used for this purpose at Great Falls as early as 1903.[2] Requirements for coal used in the smelting of copper are not rigid, although the use of coal in pulverized form brought about certain standards not necessary in the older hand-stoking practice.

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[1] A. C. Fieldner: Classification of Coal. *Trans.* A.I.M.E., Coal Div. (1930) 404–405.

[2] L. V. Bender: Development of Copper Smelting at Anaconda. *Eng. & Min. Jnl.* (1929) **128**, 301–305.

When pulverized coal is used, deposition of ash in the flues and waste-heat boilers is troublesome if the ash content is high or the ash is of such low fusion point that it does not remain suspended in the gas stream.

The increase in slag volume caused by the deposition of such ash in the charge is somewhat objectionable, although not of great importance because of the large amount of slag already present. In refining furnaces this increase in slag volume is very objectionable and coal having a low ash content and ash of high fusion point is required. Plants operating both smelting and refining furnaces purchase only one grade of coal and thus specify the coal best suited to the refining operation. Hence, coals of low ash content are preferred.

At plants where low-ash coals do not carry a premium, as on the eastern seaboard and in northern Michigan, a maximum ash content of 8 per cent is often specified. (Shipments average 6 per cent in one case and 8 per cent in another.) At Copper Cliff,[3] Ontario, coal of 7 to 10 per cent ash content is used. The Noranda smelter in western Quebec must transport coal so far that it buys low-ash (8 per cent) coal from Pennsylvania or Nova Scotia. However, high-ash coal can be used. A large company in western Canada, because of its location, found it more profitable to use a coal containing from 16 to 17 per cent ash than a more costly coal of lower ash content. A plant in western United States, situated near coal of high ash content but distant from low-ash coal, specifies a maximum limit of 16 per cent ash and charges a penalty for ash in excess of 12 per cent.

Fuel of low sulfur content is not required in the smelting of sulfide ores, because the smelted product contains sulfur. A high content of sulfur present as coarse particles of iron pyrite often causes difficulties in grinding, as the pyrite accumulates in the mill. Explosions and fires have been thought to be caused by such pyrite. At the Noranda smelter,[4] several explosions and fires resulted from the use of extremely fine (90 per cent through 200 mesh) high-sulfur (2.8 to 3.5 per cent) bituminous coal. The elimination of the pulverized-coal storage bins and the use of the direct system of grinding and firing prevented a recurrence of the fires and explosions. In the smelting of native copper ores and other sulfur-free materials, as where metallic copper is the final product, the sulfur content of the coal must be limited; usually a maximum of 1 per cent sulfur is specified.

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[3] C. F. Herrington: Canadian Nickel Smelter. *Eng. & Min. Jnl.* (1931) **132,** 219–221.

[4] W. B. Boggs and J. N. Anderson: *The Canadian Min. and Met. Bull.* 215 (March, 1930) 362–405.

The moisture content of coal used in pulverized form is important, but usually is not specified in the purchased coal, although at least one company in the East limits the moisture content in bituminous coal to 2 per cent maximum. Because the power for grinding a given coal increases with its moisture content, a low-moisture coal is preferred. In general, for efficient grinding, the higher the rank of coal the lower must be the total moisture. Bituminous coal is dried usually to 1 per cent moisture content before grinding. Thus, coal containing 1 per cent or less moisture possesses the advantage that it can be pulverized without drying. However, practice varies in this respect. In the West, at one plant a high-volatile bituminous coal is dried to 2 per cent, and at another the same grade of coal is dried to 2.75 per cent moisture content before grinding, while still another grinds a high-volatile bituminous coal containing from 2 to 6 per cent moisture and has operated successfully with as high as 8 per cent moisture content in the coal fed to the pulverizers. A company in the Southwest dries a subbituminous coal from about 16 to 3 per cent moisture content before pulverization. Lignite of still higher moisture content can be successfully pulverized and burned. Content of extraneous moisture is undoubtedly the determining factor.

Any size of coal below about 1 in. can be fed to the usual pulverizing apparatus and run of mine, slack, or one of the smaller sizes is usually purchased, depending upon the price.

The rank of coal is not ordinarily specified. Location of the plant largely determines the grade of coal used and coals ranging from subbituminous to semibituminous are successfully burned in copper reverberatory furnaces. A smelter in the Southwest uses either a bituminous or a subbituminous coal, depending upon the cost per heat unit contained, and apparently obtains equally satisfactory operating results with both grades of coal. Plants that must receive shipments by water during summer months and store coal over long periods prefer high-volatile bituminous coal. High-rank coals have the advantage of being pulverized more easily than those of lower rank, although it is not necessary to grind subbituminous coal as fine for effective combustion as it is coals of higher rank. A difference exists in the grindability of different coals of the same rank and composition. For this reason the Noranda smelter prefers Nova Scotia coal to Pennsylvania coal of the same composition.[5] For efficient burning the combustion zone must be larger for strongly coking or high-rank coals than for weakly coking or noncoking low-rank coals. Hence, other things being equal, size of furnace and amount of charge and therefore the available combustion space may determine the quality of coal to be used when a choice exists.

Copper Refining in Reverberatory Furnaces.—Copper-refining furnaces are similar in design and operation to copper-smelting furnaces and in most cases are smaller. These furnaces are used to produce refined copper from impure (blister) copper, scrap copper, or copper cathodes. In this practice, fuel quality is of more importance than in smelting practice. Ash and sulfur contents are of great importance and must be kept to a minimum. Although cheap coals of inferior quality can be utilized in reverberatory smelting, quality is more important than price in refining operations.

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[5] Reference of footnote 4.

Sulfur in the coal must be kept as low as is consistent with economy. Usually a maximum of 1 per cent sulfur is specified. Ash content is expecially important in coal used for firing refining furnaces, because it increases the amount of slag which must be retreated to recover its copper content. Also, by settling on the metal bath, the rate of heat absorption by the metal is reduced and therefore the thermal efficiency is decreased. Usual specifications stipulate a maximum of 8 per cent ash in coal used in refining furnaces. Average annual ash contents of 6 or 7 per cent are common.

At a plant in the West, the cost of smelting slag when using a subbituminous coal of 12 per cent ash content was so high that the coal was replaced by oil for firing the refining furnaces, although the coal was cheaper and was used in the smelting furnaces. Many operators believe that ash of low fusion point deposits in the slag, increasing its volume, and in flues and waste-heat boilers, whereas high-fusion ash largely passes to the stack as fly ash. Although, ordinarily, ash fusion point is not specified, coals are chosen with this factor taken into consideration. Other requirements given under reverberatory smelting apply to refining furnaces.

Copper Smelting in Blast Furnaces.—Lump or agglomerated copper and nickel ores are smelted in blast furnaces. Coke is used as fuel and part of the heat required in the smelting operation is supplied by sulfur contained in the ore. The product of the blast furnace is a matte consisting primarily of copper and iron sulfides.

A good grade of metallurgical coke of uniform size is required for copper blast-furnace practice. Sulfur content is not important. The requirements as to structure, hardness and strength are similar to those for iron blast-furnace coke, although not so rigid because of the shallow depth of charge. The size of the furnace governs requirements to some extent, as softer cokes can be used for the smaller furnaces and even the larger furnaces have been operated satisfactorily on fairly soft cokes. The size of coke used varies, depending upon location with respect to supply. It is quite likely that actual size is not important but that uniformity in size is the essential requirement. Although some operators prefer coke in sizes from 4 to 6 in. and others from 2 to 4 in., tests made at a smelter in the West were very successful with Utah by-product coke ranging from $\frac{13}{16}$ to $1\frac{1}{4}$ in. in size.

High ash content reduces the effectiveness of the coke, but ash content is not generally limited as it is in iron blast-furnace practice. Few copper blast furnaces are now operating and many of these are so distant from iron metallurgical centers that the desired quality of coke cannot be obtained.

A part of the coke charged to the copper blast furnace has been replaced by pulverized coal.[6] However, the combustion zone is so restricted that much difficulty has been entailed in burning coal in this manner. It is doubtful whether the method will achieve much use. Thoroughly dry, high-volatile, bituminous coal was found to be most satisfactory for this purpose. Coal of high ash content can be used, as Tokuzumi reports that coal containing 35 per cent ash was burned without added difficulty.

Coal and Coke in Lead Metallurgy

Lead Smelting in Blast Furnaces.—Coke is used as a fuel in lead blast furnaces. Operating conditions, and hence requirements for coke, are somewhat similar to those in copper blast-furnace practice. Sulfur is present in the charge and is discharged largely as matte; most of the lead is discharged as metal. Therefore, sulfur in the coke is not important. Ash content should be low, about 10 per cent, as it affects the cost of making and handling slag.

Size and structure of coke are important. Many operators believe that large, strong pieces are required to support the charge, which often contains large pieces of iron scrap used to keep the charge open. Coke 3 to 5 in. in size is preferred by some operators. For blast furnaces with low shafts, *i. e.*, shorter columns of charge, smaller sizes of coke are suitable. Some operators prefer coke of faster burning characteristics and of smaller size for the lead than for the copper blast furnace. They specify coke to pass a 3½-in. ring, believing that this smaller size affords better distribution in the charge and a more rapid rate of smelting. For smelting refractory high-zinc charges, Woodward[7] recommends the use of coke that passes a 4-in. ring, if it is hard and dense, or large pieces, 6 in. in size, if the coke is porous.

In the West, fast-burning coke is preferred; in some cases coke closely sized from 1¼ to 3 in. is used, and in others sizes ranging from 1³⁄₁₆ to 3 in. are preferred.

One of the prime requisites for good lead blast-furnace practice is to maintain the proper amount of carbon in the charge. This can be done best by charging coke by volume rather than by weight in order to avoid the use of too little coke when it is wet. Uniformity in size is perhaps more essential than a definite size.

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[6] M. Tokuzumi: Development of Powdered Coal Fired Copper Blast Furnace in Japan. World Engineering Congress, Tokyo (1929) **35**, 237–266.

[7] O. H. Woodward: Some Features of Lead Blast-Furnace Operations at Works of the Broken Hill Associated Smelters Proprietary, Ltd., Port Pirie. Inst. of Min. and Met. *S.A. Bull.* 321 (June, 1931).

Pulverized coal burned at the tuyeres has been used in lead blast-furnace practice to replace as much as one-third of the coke charged. Although the experimental runs were reported as successful, the process has not been successful commercially. The small combustion zone available in present furnaces prevents efficient combustion of pulverized coal in this manner. A fast-burning coal is preferred.

Lead Refining in Reverberatory Furnaces.—Pulverized coal is used in lead-refining furnaces, the furnace construction, fuel requirements and operation being similar to those of copper-refining practice.

COAL AND COKE IN ZINC METALLURGY

Zinc Smelting in Old Retort Process.—Coal or coke is used as a reducing agent for the reduction of oxidized zinc ores in externally heated retorts. Cost of fuel at the smelter, nature of the ore, and other considerations determine the kind of fuel to be used, and no definite specifications are given ordinarily. Anthracite screenings and the smaller sizes of coke are used most commonly; sometimes bituminous coal is mixed with these. The amount of ash contained in the fuel is not important, except as it affects the furnace capacity and efficiency. Although fuel of low ash content is preferred, coke screenings containing 15 to 16 per cent ash and anthracite culm containing 15 to 25 per cent ash are necessarily used in common practice.

Coal with ash of high fusion point is required when used with ore of low softening temperature; otherwise, coal with ash of low fusion point is satisfactory. The fusion point of the ash of an anthracite used in large quantities for retort zinc smelting is between 2550° and 2730° F.

Too much volatile matter in coal adversely affects condensation of the zinc, yet a small content of volatile matter is considered necessary by most operators. One grade of coke used in large quantities contains approximately 10 per cent volatile matter, although a maximum of 5 per cent is preferred. An anthracite commonly used contains between 4 and 5 per cent volatile matter, a minimum of 1 per cent and a maximum of about 5 per cent being preferred.

Reduction fuel of the lowest possible sulfur content is preferred. Most of the fuel used contains approximately 1 per cent sulfur. The size of the reduction fuel varies with the nature of the charge treated. When the ore is relatively coarse, the fuel should be quite fine.

Coal is mixed with clay and furnace residues to form a luting mixture employed in sealing off the condensers. Such coal must have sufficient caking qualities to form an adhesive and tight bond between the condenser and the retort.

For the sintering of zinc ores fuel of lowest possible ash content is required. Sulfur and volatile contents are not important. Petroleum coke is very satisfactory for this purpose and is used extensively.

Zinc Smelting in Continuous Retort Process.—In the continuous retort process briquets made from a mixture of an oxidized zinc ore and fuel for reduction are passed continuously through a vertical retort, being brought to a red heat before entering the smelting section of the retort. Low sulfur content is desirable. Ash fusibility and volatile content are of little importance. For this purpose either bituminous coal or mixtures of bituminous coal with either anthracite or coke breeze are used, depending upon costs. The type of coal and mixture of coal and coke are chosen to develop a suitable bond in the carbonized briquet. The bituminous coal used must possess good caking properties.

In another continuous retort process being developed, both the fuel and ore are sized carefully and the fuel must have a low content of ash of low fusion point.

Coal and Coke in Aluminum Metallurgy

In the electrolytic smelting process for aluminum, carbon electrodes are employed for the double purpose of conducting electricity into the bath and supplying carbon to combine with the oxygen liberated in the electrolysis. These electrodes must be of extreme purity because all the impurities present enter the bath and either affect the electrolyte or contaminate the aluminum. For example, lime and magnesia accumulate in the electrolyte, while iron oxide and silica are reduced and appear as impurities in the metal. The need for electrodes of high purity is apparent from the fact that approximately 0.75 lb. of carbon is consumed for each pound of aluminum produced. Electrodes must have high electrical conductivity and also should possess sufficient strength to permit handling. Because of its low ash and high fixed-carbon contents and other desirable physical properties, petroleum coke, which usually contains a maximum of 1 per cent ash, is widely used. The low volatile content of some pitch cokes permits their use in the manufacture of electrodes without the preliminary calcination required for petroleum coke. Anthracite[8] can be used provided it is low enough in ash. Welsh anthracite containing between 1 and 4 per cent ash is used abroad, but anthracite is not used for this purpose in the United States or Canada. To compete in quality with petroleum coke in North America, anthracite would have to contain less than 2 per cent of ash.

Summary

Requirements for coal used in reverberatory smelting of sulfide ores of copper are the usual ones for other uses of pulverized coal, as, for example, for power plants.[9] Because of location, many smelters cannot choose the most desirable coal, but for economic reasons use the most accessible coal, which may be inferior. Low ash (8 per cent maximum when possible) of high fusion point and low sulfur contents are preferred. High sulfur content is not important from a metallurgical point of view, but if present as coarse pyrite might present a fire hazard and difficulties in grinding. All grades of coal from subbituminous to semibituminous in a wide range of analyses are used.

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[8] J. D. Edwards, F. C. Frary and Z. Jeffries: The Aluminum Industry, 1, 291. New York, 1930. McGraw-Hill Book Co.

For smelting native copper ores or other products free from sulfur, coal of low sulfur content, usually 1 per cent maximum, is essential.

Coal used in pulverized form for copper-refining furnaces must be low in ash, 8 per cent maximum being customary. Ash of high fusion point is preferred. Sulfur content must be low, a maximum of 1 per cent being preferred. In coke used in copper blast-furnace practice, physical and chemical properties common to those of iron blast-furnace coke are desired,[10] although softer coke can be used and sulfur and ash contents need not be limited. For economic reasons, preferred properties cannot always be specified.

Requirements of coke for lead blast-furnace practice are similar to those for the copper blast furnace. Uniformity of size of coke is the most important requirement. Fast-burning coke is often preferred. Low ash is desirable. Sulfur content is not limited. The same quality of coal is required in lead refining as in copper refining.

In the smelting of zinc ores by the old retort process, coal or coke is used as reduction fuel. A low ash content is desired but seldom obtained. Ash of high fusion point is required for treating ores of low fusion point. A low sulfur content is preferred, 1 per cent maximum being commonly used. A minimum of 1 per cent and a maximum of 5 per cent volatile matter is preferred.

For use in sealing mixtures in zinc retort practice, coal of strong caking properties is essential. For sintering zinc ores, the lowest possible ash content is desired; sulfur and volatile matter are not important.

In zinc smelting by the continuous retort process, coal of low sulfur and low ash contents is required. Part of the reduction fuel must have strongly caking properties to bond briquets. Ash fusibility and content of volatile matter are not important.

For the manufacture of electrodes used in the smelting of aluminum, anthracite under 2 per cent in ash is required. Petroleum coke or pitch coke is used in America, but in Europe Welsh anthracite of 1 to 4 per cent ash content is used.

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[9] S. B. Flagg: Classification of Coal from the Standpoint of the Steam Power Consumer. *Trans.* A.I.M.E., Coal Div. (1930) 477–480.

[10] W. H. Blauvelt: Classification from the Standpoint of the By-product Coke Industry. *Trans.* A.I.M.E., Coal Div. (1930) 473–476.

DISCUSSION

(*H. J. Rose presiding*)

C. E. Williams (written discussion).—Mr. Eavenson has directed our attention to the paper by E. S. Bardwell.[11] In this paper, Mr. Bardwell presents an interesting comparison of the results obtained at the Anaconda smelter when using pulverized coal, oil and natural gas. While the same average daily capacity was obtained with pulverized coal as with oil, the latter gave 10 per cent greater thermal efficiency than the pulverized coal. Most of the extra heat required when using pulverized coal is thought to be due to the formation of a blanket of slag on the bath of metal; this blanket of slag results in a lower rate of heat transfer. When fired with pulverized coal, the refining furnaces produced 37 per cent more slag than when fired with oil. The ash content of the coal was about 8 per cent. Natural gas gave a lower thermal efficiency than oil. In Arizona, one plant found that about the same thermal efficiencies were obtained with pulverized coal, oil and natural gas. Much depends upon the type of installation and operation.

In another paper on this subject,[12] Mr. Bardwell stated that the Anaconda company found it more economical to use coal containing 8 per cent ash and 1 per cent sulfur from Utah and Wyoming than coal from the Great Falls, Mont., field containing 22 per cent ash and 4 per cent sulfur, in spite of the much shorter haul and lower cost of the Montana coal.

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[11] E. S. Bardwell: A Comparison of the Use of Various Fuels in Copper-refining Furnaces. A. I. M. E. *Tech. Pub.* 457 (1932).

[12] E. S. Bardwell: Pulverized Coal as Fuel for Copper-refining Furnaces. *Trans.* A. I. M. E. (1926) **74**, 592.

Recent Research on Ground Movement Effects in Coal Mines and on the Strength of Coal and Roof Supports*

COMPARATIVE METHODS USED IN DIFFERENT EUROPEAN COUNTRIES AND THE UNITED STATES

BY GEORGE S. RICE,† WASHINGTON, D. C.

(New York Meeting, February, 1932)

THE increasing use of mechanization at the face of the workings in coal mining and the consequent necessity of special supports of the roof has led, in several countries, to considerable scientific investigation, including physical tests in mines as well as on specimens of roof rocks to obtain data on the behavior of the immediate roof strata and on the effect of its pressure on supports while the coal is being mined.

The investigations discussed in this paper do not include studies of the ultimate effect of subsidence and movement of the roof rock over the mine excavation, upon higher strata and the surface, but are confined to the immediate underground effects in the workings. The more recent principal investigations which have been brought to the attention of the writer are as follows:

Germany.—The trial of strip packing, the usual method employed in Great Britain in longwall workings, as compared with complete packing heretofore used in Germany and in France to minimize subsidence; the behavior and strength of metal props under load; the testing of the strength of coal measure rocks, in the laboratory of the Technical High School of Breslau; and an investigation, by a special commission, concerning "bumps" as a possible cause of instantaneous outbursts of gas in coal mines of Lower Silesia.

France.—The cause of instantaneous outbursts of gas in the Gard mines and tests by the official French experimental organization of the "electrical resistivity" of coal as a possible means of detecting a condition preceding an instantaneous outburst.

Great Britain.—The subsidence of the roof and rise of the floor preceding and during excavation of the coal, in longwall and in room-and-pillar or bord-and-pillar workings; the compression and lateral movement of the road packs in longwall back from the face; the extent of separation of the immediate roof layers from the higher strata following mining excavation;

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the pressure on props and supports; the effect in longwall workings of different arrangements of pack-walls and strip packing; and in the physical testing laboratory, the strength and efficiency of steel props of various kinds at the face and of arches in the roadways.

United States.—The strength of coal and rock specimens; the compressibility and unit bearing strength of coal-in-place, horizontally and vertically, in the Bruceton Experimental Mine; the study of different systems of timbering carried on in various coal fields by field engineers of the Bureau of Mines in cooperation with mining companies; the study of subsidence of the roof carried on by mining companies and also in cooperation with engineers of the Bureau; the study of the character and flow of gases into mine workings.

INFLUENCE OF MINING CONDITIONS ON CHARACTER OF INVESTIGATION

The character of an investigation necessarily is influenced by the system of mining, and this in turn is affected by the physical characteristics surrounding the coal bed in the particular field for which the investigation is carried on. In European fields the coals now being mined are generally deep, ranging from about 1200 to 4000 ft. Moreover, the coal-measure rocks are predominantly shaly and weak, generally shear or break in tension easily, and at great depth may flow under the pressure. Also, the coal beds mined in most coal fields of Europe are, on the whole, thinner than those mined in the United States. Exceptionally, there are thick beds, as in the Upper Silesian field of Germany and Poland and in South Staffordshire, England.

While the area of the individual coal basins of Europe is small, compared with those of North America, the aggregate thickness of workable coal beds is great and the beds usually are in densely populated regions, so the effect of subsidence of the surface is often very important and this influences the importance of thorough packing of the mining excavations.

The coal now mined in the United States is at a shallower depth, generally less than 1000 ft. from the surface and surface subsidence is usually less important; also the coal-measure rocks usually contain more sandstone and limestone strata, hence are stronger, and the individual coal beds have a greater average thickness than European coal beds, and are further apart, but there are many exceptions to these generalities in the coal fields of either continent.

These differences in conditions indicate the suitability of the method of "longwall advancing" with packing of the goave, and the unsuitability of pillar workings for most European mines. On the contrary, longwall advancing in most American mines is uneconomic or impracticable because of lack of suitable packing material, whereas pillar workings are, in general, admirably suited to the conditions, especially where mechanization is employed. Formerly, as is well known, the method of pillar work used in the United States, by the primitive room-and-pillar system, was very wasteful of coal reserves, as pillars were often not extracted. Studies made under the supervision of the writer, for the U. S. Coal Commission, indicated that as late as 1922, in mining operations in this country, an average of one-third of the coal originally in a unit of area was lost or permanently left underground.

During the last few years there has been an increasing improvement in the mining methods and rapid extension of mechanization. Under the system now used by many companies for extracting pillars, less than 20 per cent of the coal may be taken out in narrow advance work, permitting the extraction of most of the coal on the retreat, by employing a long retreating break line. Under the best practice an ultimate extraction of 90 to 95 per cent of workable coal is obtained.

In a few districts thin coal is mined by the longwall advancing method, making practically complete extraction, but this is the exception, as under most mining and labor conditions in the United States the cost is greater than by pillar methods.

Retreating longwall and likewise pillar extraction in a long, more or less straight break line, cause effects on roof subsidence similar although not identical to those caused by longwall advancing; hence the value to American engineers of the studies now being conducted in certain mines of the principal coal-mining countries whereby the strength of coal and rock and mine supports—props, arches, etc., are tested in laboratories and underground in the mine faces.

Classification of Research Work

The ways in which the investigations are being carried out may be classified as follows:

1. Laboratory tests of the bearing strength and mode of rupture of small specimens of coal and coal measure rock.
2. Laboratory tests of the strength and compression of wood and steel props, cribs, beams, arches and other artificial roof supports.
3. Underground testing of unit roof pressure, movements of the roof, floor, and sides of the face working places and passageways, and also the loading and deflection of artificial supports.
4. Underground testing of the elasticity, plasticity, and flow under load of coal-in-place, when acting as a buttress for stoppings and hydraulic dams, and determination of its safe unit bearing strength, parallel and perpendicular to the coal bed.
5. Underground studies and tests of the loads carried and deflection of different kinds of artificial roof supports in the face workings, such as steel props and arches, and wood props and cross-timbers; also their spacing and arrangement.

6. Underground studies and trial tests of arrangements in longwall working of pack-walls, strip packing; also partial and complete stowing or gobbing by hand, by hydraulic means, by pneumatic agency, and by mechanical stowers.

7. Underground studies in headings or entries and tests by trial of steel arches, massive concrete arching, and concrete block and brick arching.

8. Testing the electrical resistivity of coal in the laboratory and in the face of workings to determine if possible whether there is a difference in resistivity of coal in normal condition from that in an area subject to instantaneous outbursts of gas with a view to getting warning of impending outbursts with advance of a face or heading.

INVESTIGATIONS ON SUPPORT OF MINE WORKINGS

In a brief paper it is impossible to give complete details of the numerous investigations that have been made or even of those recently undertaken. A general review only can be undertaken. During the past two years an unusual number of intensive studies, some accompanied by physical tests, have been begun in the principal coal-mining countries, stimulated, on the one hand, by increasing mechanization in the face workings, which has introduced new hazards; and on the other hand, by an intensive effort to lessen the number of accidents to miners from falls of roof and coal.

Laboratory Tests of the Bearing Strength of Small Specimens of Coal and Coal Measure Rocks in Hydraulic Pressure Machines.—This was formerly the only method of testing and has been employed extensively in various countries. The test data have been employed until lately in deducing the necessary size of coal pillars for support of overlying strata and the mode of rupture of the roof rock. As described in mining literature tests of the crushing strength of small blocks and prisms of anthracite were made in 1900 for the Scranton Engineers Club by Cornell and Lehigh universities and Pennsylvania State College.[1] Other tests were made at Lehigh University in 1907.

Tests on the crushing strength of bituminous coal were made in 1906 by Armstrong College, Newcastle-on-Tyne, England; at the University of Illinois, and more recently by the School of Mines, India (1928), and by West Virginia University (1932).[2]

All of the foregoing testing was done with small specimens of coal in the form of cubes or prisms of 2 to 6 in. diameter of cross-section with four free faces. The unit crushing strength determinations in these various tests were extremely variable, depending on (1) the character of the coal, Pennsylvania anthracite giving the highest values for small cubes up to 8600 lb. pressure per square inch, and bituminous coal from about 4000 down to 1000 lb.; and (2) the size of the specimen, the larger the cube or the higher the prism, the lower the crushing strength per square inch of loading surface.

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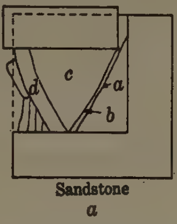
[1] W. Griffith and E. T. Conner: Mining Conditions under the City of Scranton. U. S. Bur. Mines *Bull.* 25 (1912). See Appendix, 77.

[2] C. E. Lawall and C. T. Holland: Some Physical Characteristics of West Virginia Coals. See page 100.

In 1914 the U. S. Bureau of Mines carried on a series of tests of cubes of bituminous coal (Pittsburgh bed) of various sizes up to a 4½-ft. cube, in a 10,000,000-lb. press and found that the unit crushing strength rapidly decreased with increase in size of cube, over 4-in. size, manifestly due to the presence in the larger specimens of planes of weakness—face and butt cleats. This led to the conclusion of the writer that the data obtained by this type of testing had little or no value for practical mining application.[3]



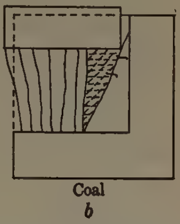


FIG. 1*a*.—BREAKS IN SANDSTONE SPECIMEN IN CRUSHING MACHINE. (*Müller.*)

FIG. 1*b*.—BREAKS IN COAL SPECIMEN IN CRUSHING MACHINE. (*Müller.*)

Müller Laboratory Testing.—A new method of laboratory testing has recently been employed by O. Müller of the Technical High School, Breslau, Germany, in which he has tested specimens of coal and coal measure rock partly confined in a steel form, leaving only one free side. In his report[4] he says: "Although the opinion of different investigators differs on the magnitude and direction of pressures existing in mines, yet they are unanimous on one point—that geologic structure as well as the method of working a mine are determining factors." Müller points out that in the testing of rocks there are certain names for their physical properties, such as "solid, plastic, brittle, and elastic." He also comments: "To what extent the results of experiments conducted in a laboratory, where only small scale testing is possible, can be compared with the effects of dynamic movements in large masses of rock remains unanswered at the present time." However, he says that "an opinion exists that in the testing of materials where particular attention is paid to reproducing actual conditions as much as possible, the absolute values obtained are correspondingly correct." He remarks, however, that the factor of time which is so important in natural phenomena is almost impossible to allow for in small scale tests.

Dr. Müller divided his experiments in five different groups:

1. Resistance tests to pressure (crushing tests).

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[3] Tests of Strength of Roof Supports Used in Anthracite Mines of Pennsylvania. Report of Anthracite Mine Cave Commission; introductory review by George S. Rice, U. S. Bur. Mines *Bull.* 303, 7.

[4] *Glückauf* (Nov. 22, 29, 1930).

2. Fracturing tests under conditions which he considered to be similar to natural ones.
3. Plasticity tests of the rock specimens.
4. Elastic property tests of the rock specimens.
5. Compression tests.

The material tested—coal, shale, and sandstone—came from the Upper Silesian coal measures.

Testing under the first method was conducted with a 100-ton press used for testing building materials. In certain cases pressure was applied both parallel and perpendicular to the plane of cleavage.

The specimens were usually in the form of a cube but sometimes had the form of a cylinder, and were cut out by a rock saw or diamond drill. The cubes were usually 10 cm. (4 in.) on the side; the cylinders had a diameter of 40 mm. and were 45 mm. high (1.6 by 1.8 inches).

After the top and bottom of the specimens had been cut and ground smooth, they were placed between the plates of the press. This series of tests on the resistance to pressure was therefore conducted in the usual manner for crushing strength tests, with opportunity for breaking out on four sides. The average maximum strength of four specimens was as follows:

Sandstone, perpendicular to the stratification, 11,800 lb. per sq. in., and parallel to the stratification, 11,300 pounds.

Shale, according to previous work by Cordebas, quoted by Müller, found a variation from 1200 lb. with soft shale to 4400 lb. for extremely hard shale, but Müller obtained an average for six specimens he tested, of 12,000 lb. per sq. in. perpendicular to the bedding and for four specimens 8250 lb. per sq. in. parallel to the bedding.

Coal (bituminous), Müller found, had a wide range in the maximum strength of coal specimens from two beds in Upper Silesia, ranging from 1900 to 4300 lb. per sq. in. in 70 cubes from the Heinitz bed, with an average of 3100 lb., and an average of six cubes from the Schuckmann bed of 2400 pounds.

In the rupture of most of the cubes of rock specimens an explosion-like effect occurred when the maximum pressure was reached. The specimens in failing always showed diagonal fractures producing opposing pyramidal shapes in the case of cubical specimens and conical in the case of round cylindrical specimens.

In the second method of testing, which Dr. Müller considers duplicates as nearly as possible conditions in mines and which is a novel method of laboratory testing, the specimen cubes were placed in a steel form with one side open and a pressure die at the top exactly fitting the form (Fig. 1). The open side is supposed to represent the working face or side of a gallery since the only escape from a crushing pressure is the open side, as it would be from a coal face or side of a heading. The cubes thus tested were of a size identical to that in the previous tests—10 cm. (4 in.), fitting tightly within the steel shell. Any small spaces were filled with plaster of Paris. The pressure on a sandstone or shale specimen was slowly increased until a sharp report was heard and pieces were thrown off at the free face, similar to the case of rock-bursts in tunnel work and stopes. The first fracture plane is smooth and, because it is not covered with dust, Müller concludes that the bursting resulted because the tensile strength of the rock was exceeded and that the fracture plane was not due to shear effects. The load produced by the machine drops immediately when the rupture of a specimen occurs. If the loading is again put on by lowering of the pressure plate it does not reach the previously attained maximum but results in the partly crushed cube further decreasing in height with increase in load and the fractured parts and small pieces are thrust toward the open face. When the height of certain specimens was diminished 4 cm., equal to 40 per cent of the original height, the respective tests were discontinued and the fractures examined and photographed. (Figs. 1*a* and *b*, diagrams of the partly crushed specimens, are reproductions of Figs. 14 and 15 of Müller's paper.)

The pressure tests of coal specimens in the steel form with one open side were accompanied by effects very similar to those in rock and shale testing, sudden bursting with a sharp report and the formation of fracture planes. Fig. 1*b* indicates numerous fracture planes in the coal specimen, from the central part to the open side, which are more or less vertical with a zone of wedge-shaped crushed coal against an inclined plane at the back, whereas Fig. 1*a* of the sandstone rupturing, if correctly portraying average effect, shows that there is a large wedge, with edge downward which has moved on an inclined plane at the back. This inclined fracture appears to be a shear plane.

Although, as Müller acknowledges, it is difficult to say whether similar conditions and effects occur in mines, it appears to the writer that the true bearing strength of coal and coal measure rock in the mine is more nearly obtained by this laboratory method than by any previous laboratory method. It is suggested that it would be valuable to obtain comparative results in larger apparatus of this kind.

The third method of Müller to determine plasticity was to put specimens embedded in lead in thick copper shells placed in a pressure machine, and load to point of rupture of the specimen. This is a method used with variations in detail by many previous investigators. Müller obtained results as regards the character of fracturing but does not record quantitative results as to pressure effects. He then employed another way, after the method of Kick who experimented with marble. The specimen was put into a cubical steel form with tight-fitting pressure die and a coin placed under the die. The pressure was increased until the coin was forced into the specimen. Shale and coal were tested by this method.

It required 10,600 lb. per sq. in. to do this with slate and 31,000 lb. for coal, from which Müller reasons it is more difficult to force coal into a plastic condition than slate. This reported test result indicating a low order of plasticity of coal seems very dubious to the writer to apply to practical mining conditions from results of tests on coal-in-place at the Bruceton Experimental Mine.[5] Müller, however, reaches one conclusion which will be generally accepted, that the crushing strength of rock at depth may increase several fold when some side pressure exists as compared to when there is no side restraint.

Müller's fourth group of tests made on prisms relates to the determination of elasticity of coal and coal-measure rocks. His findings confirm those of other investigators, that rocks and hard shale are more or less elastic and when the load is increased and subsequently is decreased there is a noticeable hysteresis loop shown in plotting the curves (as illustrated by graphs in his paper). He found as might be expected a much greater proportionate decrease caused in the permanent length of coal prisms than of the rock prisms.

As concerns the elasticity of sandstones and slate, Müller says: "Rocks at depth have to bear considerable side pressure which increases their elasticity, particularly through movement caused by mining excavation. Further, since in all probability the sandstone will be considerably more elastic at depth than the shale, the curve of E-values for the sandstone is steeper (as shown in his graph) than the E-curve for the shale which is continually decreasing."

This leads to his concluding that the elastic force of the sandstone when released will have several times greater effect than is the case with slate; therefore, there will be sudden liberation of large amounts of energy from the sandstone as mining excavations approach, that may produce effects such as, for example, bumps and outbursts which have to be attributed to its high elasticity.

Müller's fifth group of experiments was to determine the compressibility of the rocks and coal as related to the stored energy. Müller says that the first one to discover its importance was Langecker. A pressure arrangement was chosen with two main pressures being equal and a third pressure which was much higher. "From theoretical consideration it was determined that the back-pressure when working a mine is at least 100 kg. per sq. cm. (1470 lb. per sq. in.)."[6] The compressibility was determined in testing cylinders.

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[5] G. S. Rice, H. P. Greenwald, H. C. Howarth and S. Avins: Concrete Stoppings in Coal Mines for Resisting Explosions. Bur. Mines *Bull.* 345 (1931) 59; also, Bur. Mines *Tech. Paper* 527, Compressibility and Bearing Strength of Pittsburgh Coal Bed—First Tests of Lateral Compression with Total Loads up to 500 Tons.

[6] Müller's meaning of back-pressure is not clear unless he refers to the ground static pressure which would be about 1470 lb. per sq. in. at 1300 ft. deep.

The following compression values were obtained:

Pressure		Total Compression Obtained in Per Cent		
Kg. per Sq. Cm.	Lb. per Sq. In.	Sandstone	Shale	Coal
200	2,940	0.19	0.16	0.63
300	4,410	0.27	0.20	0.83
400	5,880	0.30	0.27	1.08
500	7,350	0.40	0.32	1.25
1,000	14,700	0.57	0.71	2.22

Müller comments: "The main result, as indicated by the above figures, is the fact that the coal-measure rocks show a total compression of more than 0.1 per cent with an axial pressure as low as 100 kg. per sq. cm. (1470 lb. per sq. in.) and a side-pressure of less than 100 kg. per sq. cm. He adds, "This agrees with the calculated figures but is incomprehensible."

Müller, in his final conclusions, agrees with Lindermann and certain other German investigators that the cause of rock bursts and in coal mines of instantaneous outbursts of gas is the stored energy in the rocks, and particularly in sandstones.

The experience of the writer indicates that this may be true as regards some kinds of "bumps" but he does not agree that instantaneous outbursts of gas can be attributed to this cause, unless the gas is already stored and held in so-called "nests" caused by previous tectonic movement that crushed the coal; "bumps" (equivalent to what are termed rock-bursts in metal mines) have occurred in coal mines in many districts without notable outbursts of gas, but instantaneous outbursts of gas have occurred only under exceptional geologic conditions in comparatively few mining districts.[7] Violent bumps are liable to occur in any coal field where there are thick strong beds of rock overlying coal and unsuitable methods of mining are employed.

LABORATORY MODEL TESTING METHODS

The testing of miniature models of rock strata has been extensively employed in studying the geotectonic folding of rocks to determine the character of stresses, flows and faults produced thereby, and also in determining, in engineering constructions, the character of strains and deformations under load. It has been used less frequently for studying ground movement resulting from mining because of the difficulty of duplicating the compressive and tensile strength of strata on a small scale. Recently this problem has been attacked in an entirely new way, by Prof. P. B. Bucky of Columbia University. This employs Newton's

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[7] See papers and discussion on Instantaneous Outbursts of Gas by G. S. Rice, P. A. C. Wilson, B. Caufield and others in *Trans.* A.I.M.E., Coal Div. (1931) 75–136.

"theory of similarity," as a basis for the testing of models of the rock strata in proportionate size in relation to the mining excavation. The testing of the models is done in a centrifuge apparatus which, by the rapidity of rotation, produces dynamic similarity in the model to the effect of weight of the rock strata. Bucky has given a detailed description with examples of tested models and the underlying theory.[8]

Mathematically, the theory of this novel method of laboratory testing has been accepted by competent judges of structural testing methods. Its successful application to specific mining problems of ground movement appears to be principally dependent upon proper construction of models in exact proportions to the original rocks which may be intersected by joints and slips, say as 1 to 1000 or 1 to 2000, as the case may be. While the effect of gravity is simulated by the centrifugal force, this acts radially and is at right angles to the specimen tested only at its center: with a centrifuge of larger diameter proportionate to the model, which Professor Bucky proposes to construct, this may be a less important factor. There appear, however, to be other difficulties in attempting to duplicate, in a vertical section of strata, the effect of lateral restraint of adjacent strata that obtains in mining at depth and arching stresses in contiguous broken ground; also the effects of sliding on dipping beds, contact planes, or on inclined fault planes. Further developments of this novel method, as applied to ground movement and roof stresses of mining excavations with the use of larger centrifuge apparatus which has been constructed and now is being used in testing by Professor Bucky, will be watched with great interest by all interested in the problems caused by mining excavations.

Laboratory Testing of Roof Supports

In the course of an investigation by William Griffith and Eli T. Conner into "Mining Conditions under the City of Scranton, Pennsylvania" in 1911, tests were made by Lehigh University on different kinds of supports used in the anthracite mines.[9]

Large scale tests of roof supports were made at Pittsburgh, Pa., with a 10,000,000-lb. capacity hydraulic press, by the U. S. Bureau of Mines and Bureau of Standards for the Pennsylvania Mine Cave Commission, in 1912. The results are given in a report of the Commission to the state legislature. Its report was subsequently published[10] by the U. S. Bureau of Mines. The writer knows of no extensive laboratory testing of coal mine roof supports made in the United States since these investigations.

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[8] P. B. Bucky: The Use of Models for the Study of Mining Problems. A.I.M.E. *Tech. Pub.* 425 (1931).

[9] Reference of footnote 1.

[10] Reference of footnote 3.

TESTS OF WOOD PIT PROPS BY THE SAFETY IN MINES RESEARCH BOARD, ENGLAND

Extensive laboratory testing of props or posts, both of timber and steel, have been conducted recently under the Safety in Mines Research Board, by the Support of Workings in Mines Committee of that Board. The testing work has been done at the Imperial College of Science and Technology, London. The tests on the wood props were carried on from 1925 to 1930 and on steel props between 1927 and 1929. The report[11] on the former was issued in 1930 and the report[12] on the latter in 1931.

The strength of the timber props was found to vary greatly although in the respective groups they were practically alike in size. Also, it was found that the effect of moisture on the strength of timber is very important: room-dry timber is about 50 per cent stronger than saturated timber. The tests also demonstrated that the longitudinal compression of the props under maximum load before failure is small.

TESTS OF STEEL PIT PROPS BY THE SAFETY IN MINES RESEARCH BOARD, ENGLAND

Steel props were first used extensively in Germany, although a combination of wood prop with steel sleeve or pipe casing has been used successfully for more than 13 years, from personal observation of the writer, at the Newbattle Colliery, Scotland.

Steel pipes have been little used in the United States because of the predominance of room-and-pillar methods, as under this method in the past room pillars were either not pulled or else stood for years before pulling and hence the props or posts must stay for a long time and are more or less buried by the waste material gobbed or by falls of roof. Steel props are much more expensive than wood and can only be used economically when they are drawn out regularly and the roof allowed to cave to the next line of props.

During the last few years, the growth of mechanical loading at the face of the mine and the increasing use of retreating long faces with caving of the roof behind by drawing props in some cases, has led to the trial use in some mines of special steel props and there is increasing interest in the practice abroad, especially in the props which may be easily collapsed when necessary to pull them.

The development of steel props has been toward props that will yield gradually under a predetermined maximum load yet sustain a lesser load like soft wooden props as the roof gradually lowers in longwall or in retreating pillar extraction line. Very rigid props might break the roof or be ruptured.

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[11] S. M. Dixon and M. A. Hogan: Tests on Timber Pit Props. Safety in Mines Research Board *Paper* No. 72 (1931).

[12] T. Ashley, S. W. Dixon and M. A. Hogan: Steel Pit Props. Safety in Mines Research Board *Paper* No. 58 (1930).

The natural tendency is first toward the jack-screw prop which permits gradual shortening at intervals to prevent excessive load which would rupture the roof or cause the prop to fail. Jack-screw props are not new in principal, the writer in 1892 designed and tried some in a longwall face in the Ladd mine in northern Illinois as legs of cross bars: the latter were steel rail girders and were held by iron shoes on the heads of heavy jack-screws. They were used to protect a movable trackway along the face in connection with a longwall mining machine then in its first stage of development. While the steel roof supports were promising, various conditions, an unusually tender roof, and the usual difficulties of pioneering, including troubles with the development of a longwall cutting machine prevented complete success at that time, although the same principle was employed in West Virginia and Pennsylvania in recent years and is being successfully used in certain European mines.

The later efforts of European practice in the development of steel props has been to provide friction-slipping devices containing wood or steel wedges so the prop may yield gradually under a maximum load, yet sustain a lesser load as the roof subsides or the floor heaves; or else to use rigid pipe props with timber footings; or a third type, a semirigid prop like the Newbattle device of having the wood prop inside of the pipe casing extend above or below the steel casing.

The Safety in Mines Research Board in 1928–9 carried on a most valuable series of tests on the principal types of steel or combination wood and steel props used in Great Britain, Germany and France, the results of which were published in 1930. The 13 props tested are illustrated in Fig. 2 and the detailed figures are given in *Paper* No. 58.[13]

From these tests it is evident that the ultimate strength of the combination steel props, now more or less used in European coal mines, is little if any greater than good wooden props used in Europe and undoubtedly is much less than the hard wood props—oak and Georgia pine—of the United States. Further, strong wood props compress little under load; less than most of the combination steel props. The greatest advantage of steel props appears to be in advancing long faces or longwall with recovery of the props. In this work it is manifest that the ideal steel prop is one that will slowly but regularly yield with uniform resistance. That none of the steel props tested will meet this ideal is evident from the load-shortening curves shown in Fig. 3 reproduced from the British report. The best curve for high resistance is shown by the Tait prop (6) and for low resistance by the S.A.M. prop (10) and the Mureaux (11).

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[13] Reference of footnote 12.

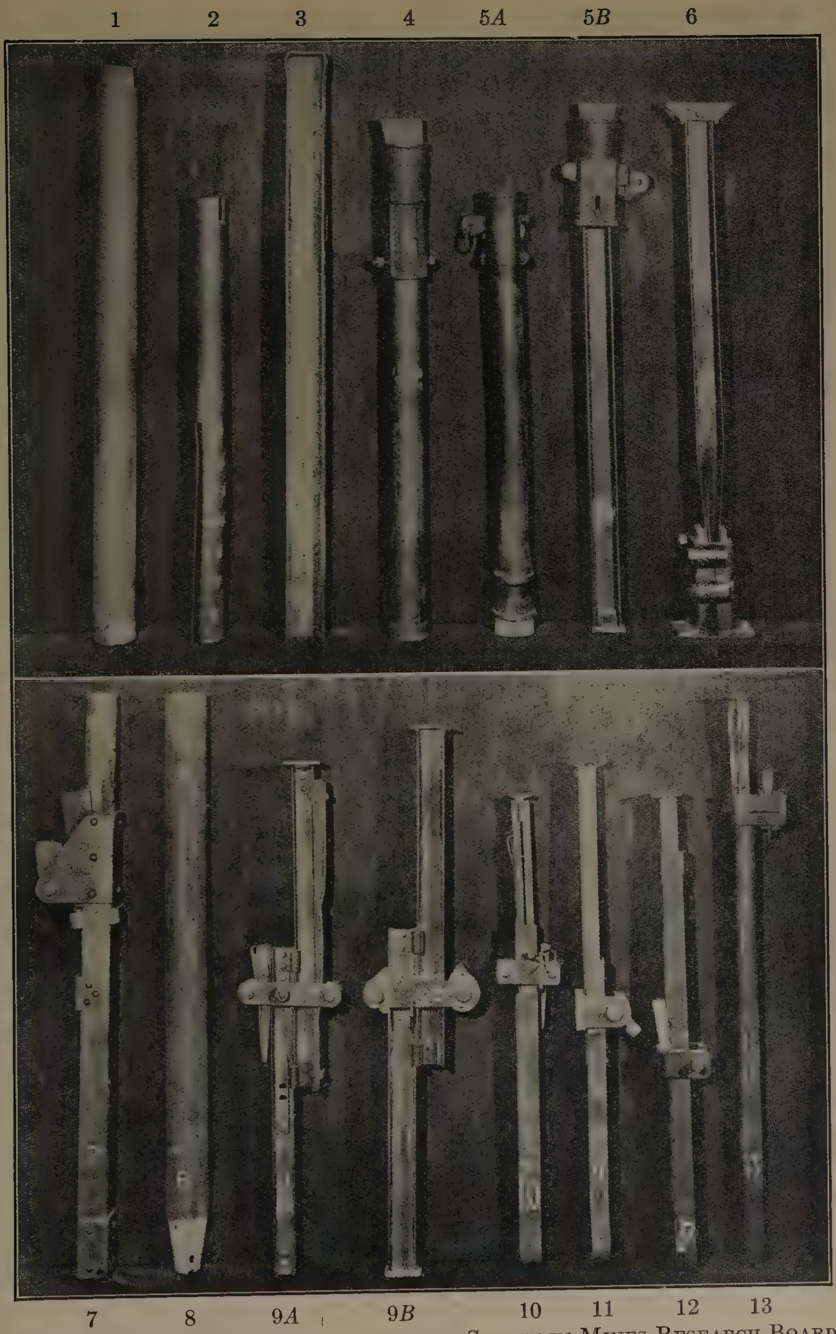

Fig. 2.—Wood and steel props tested by Safety in Mines Research Board.

Upper row: 1, Steel tube, with timber filling; 2, Chatham; 3, steel joist; 4, Butterley; 5A, S.F. (tubular pattern); 5B, S.F. (joist pattern); 6, Tait.

Lower row: 7, Berrisford; 8, steel tube with tapered wood plug; 9A, Sarre; 9B, New Sarre; 10, S.A.M.; 11, Mureaux; 12, Schwarz; 13, Ruhr.

A type of steel prop not included in this series of tests at the Imperial College (London) is a plain seamless pipe in combination with a wood "lid" or cap piece and a thick soft wood footing which cushions and provides the necessary "yield" as the roof descends and the floor rises in a longwall face. It is being experimented with in connection with tests being conducted in a Scottish colliery, by Dr. A. Winstanley of the Safety in Mines Research Board staff. From observation of the writer, this form of steel prop in combination with a wood cap piece and wood cushion below appears simple and effective.

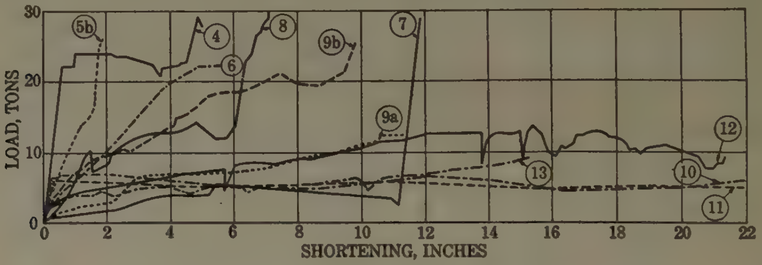

FIG. 3.—PROP LOADS AND SHORTENING CURVES. (*Redrawn from Fig.* 3 *in Safety in Mines Research Board Paper No.* 58.)
4, Butterley prop; 5b, S.F. prop (joist pattern); 6, Tait prop; 7, Berrisford prop; 8, steel tube (with tapered wood plug); 9a, Sarre prop; 9b, new Sarre prop; 10, S.A.M. prop; 11, Mureaux prop; 12, Schwarz prop; 13, Ruhr prop.

TESTS OF STEEL ARCHES

Steel arches are now extensively employed in European coal mines in temporary as well as in more permanent roadways as a substitute for wood, brick and concrete. They have been tested for strength as an arch in the laboratory of the Imperial College, London. The reports on the tests have been given rather briefly in the annual reports of the Safety in Mines Research Board. In the Sixth report (1927) on page 40; in the Eighth report (1929) on page 44; and the Ninth report (1930) on page 51.

METHOD OF USING STEEL ARCHES IN MINES

A paper, "Five Years' Experience with Steel Arches as Roadway Supports,"[14] describes results obtained in a North Staffordshire mine where about 12 miles of roadway are supported by steel arches and 1000 tons of arches used annually. The authors state: "When a large district which had been temporarily closed in 1924 was reopened in 1927, the timbered lengths were a succession of heavy falls, whilst in the steel-arched lengths only the rails required levelling in order to make them ready for haulage purposes."

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[14] R. Bennett and M. B. Gardner: *Trans.* Inst. Min. Engrs. (1928) **75,** 93.

The special merit of the steel arches is the effect of putting broken blocks of roof in arch form, so the principal load is probably carried by the interlocking blocks rather than the arches. A further merit, that effects economy, is that if arches are deformed they can be taken out, straightened and reused if not too much crumpled.

Another type of arch—a cambered arch—is extensively used in Lancashire and North Staffordshire mines where the roadway has "climbed" into the original roof. This is the natural tendency in longwall mines from heave of the floor and brushing from time to time of the roof. Where the ribs of the passageway are strong, the cambered arch can be advantageously used.

Underground Testing of Roof Pressure and Movement of Roof and Floor

The most extensive testing in the face workings of coal mines as yet undertaken is that which has been carried on for several years by the Safety in Mines Research Board, Great Britain, in cooperation with a committee of several mining institutes and also with individual coal-mining companies.

This testing had followed an extensive program of field investigations, which is still continuing, of the Support of Workings in Mines Committee of the Board, in different districts.

Falls of ground—roof or coal—in every mining country cause the largest single class of accidents, totaling 40 to 50 per cent of all mine accidents. Six papers on the "Support of Workings" in six different mining districts of Great Britain, have been published by the Board.

Reports on the testing work conducted in mine workings have not yet been published by the Board except briefly in its annual reports, but recent informal papers have been given before mining institutes by the respective research engineers. Two of the more detailed informal reports appear in the *Transactions* of the Institution of Mining Engineers.[15]

These investigations have undertaken to measure instrumentally (*a*) the extent of movements on a time basis, of the roof and floor in longwall and (*b*) in headings in the solid coal in bord and pillar workings, as mining excavation proceeds; (*c*) in longwall work, the compression of the pack-walls; (*d*) the compression, also on a time basis, of the coal-in-place as mining approaches; (*e*) the load carried by props, and (*f*) the opening of spaces between rock stratum over excavations. In addition, general observation has been made on the effect of different arrangements of props, cross-bars or roof straps, of spacing, thickness, and character of pack-walls, also a study of roof-breaks and their position, direction, and angle of pitch particularly in longwall workings.

The instruments employed in these investigations are dynamometer props for determining the load carried by each at the given point and at periodic time intervals and the subsidence of roof and rise of floor during the cutting and removal of the coal at the face. Dr. Winstanley, in his investigations, has employed for dynamometer props, steel bars with shrunk-on collars. Mr. Walker, in his testing in bord-and-pillar workings in a Durham Colliery, has employed the Wazau dynamometer prop which is described in the Sixth Annual Report (1927, page 39) of the Safety in Mines Research Board, essentially as follows: The dynamometer part consists of a steel cylinder with a steel plate fastened to it to form a thin but broad chamber. This is filled with mercury and when the cylinder is put under load its lateral dilation and longitudinal contraction reduce the volume of the mercury chamber causing some mercury to be expelled. The volume of mercury expelled is measured by means of a micrometer screw and a glass index tube.

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[15] A. Winstanley: Longwall Roof Control, and A. Walker: First Progress Report of an Investigation into the Causes of Falls, and Accidents Due to Falls in Bord-and-pillar Whole Workings. *Trans.* Inst. Min. Engrs. (July, 1931) **81**, 373, 427.

In the Seventh Annual Report of the Board (1928, pages 61 and 62), the Eighth (1929, pages 43 and 44), and the Ninth (1930, page 48) brief descriptions are given of the method of underground testing with the props. One prop was under test for 106 days while the face advanced 24 yards. It was found that as a rule the load on a dynamometer prop first set near the face increased even if the face was not advanced. In one case a load on a dynamometer prop increased to 32 tons, but as a rule the loads on the props were not over 15 tons.

In the Walker tests the dynamometer props were set under planks or cross-bars which in some instances were steel corrugated straps. Such steel straps are now much used for both roof planks and lagging, in face workings in British mines in place of wood planks.

The subsidence of roof and raise of floor at the points of placing the dynamometer props is measured by a subsidence self-recording gage constructed in the Safety in Mines Research Board instrument shop at Sheffield. This is a telescopic tube with spring coil to hold firmly against plugs inserted in holes in the roof and floor. The movement caused by convergence of the roof and floor is recorded by a lead point tracing a line on paper placed on a rotary drum actuated by clockwork. The record is removed and a new paper placed on the drum each 24 hr., or once a week in a different model, thus giving a continuous record of the amount of subsidence.

In some of the investigations bars were placed in the side pack-walls, the ends projecting, at horizontal intervals in the roadway passage, and also plugs were placed in the rib made by brushing the roof above the pack-walls, also in the floor below. Then by leveling from some bench mark in undisturbed ground, the relative subsidence of roof and lift of floor could be determined and also the squeezing in of the sides of the passage by measuring across between opposing ends of the bars set in the pack-walls.

Other observations have been made in longwall work by instruments such as: a micrometer caliper gage for measuring change in height between roof and floor and a special clinometer on a bar placed against knife edges in plugs which are set in the roof and floor in the roadway passage for determining respectively, as the face advances, the change of inclination of the roof and floor. The change in horizontal length of the surface of the roof stratum in compression or in tension, as the case may be, is determined by placing plugs in the roof at certain distances apart and measuring at time intervals with a micrometer gage. It is found that the roof in longwall is first in compression as it cantilevers over the face, then as the face advances it is in tension and "breaks" occur parallel to the face. The same condition would probably be true in extracting pillars. Diagrams showing subsidence and deflection of roof and rise of floor, also the changing load on dynamometer props in certain longwall mines in

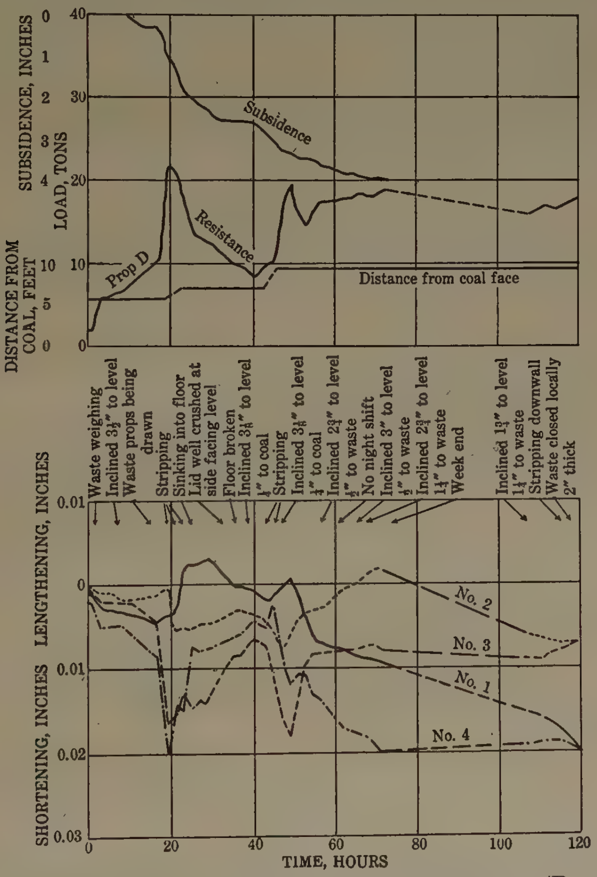

FIG. 4.—LOAD AND SUBSIDENCE RECORD OF A DYNAMOMETER PROP (D SHOWN IN PLAN OF WORKINGS).

Scotland, illustrate the paper[16] by Dr. Winstanley. Fig. 4 (reproduced from his Fig. 6) represents one case of his recorded results of roof subsidence and load carried by dynamometer props.

Some tentative conclusions are presented in the paper as to the way the roof behaves with different kinds of pack-walls but as testing is continuing it is not advisable now to draw final conclusions. Interesting instrumental evidence is presented showing that subsidence of the roof accelerates at a rapid rate: (1) when the machine in cutting reaches a point opposite the subsidence recording instrument; (2) when stripping and loading of the undercut coal is being done opposite the instrument, and (3) when the back line of props opposite the subsidence instrument is being withdrawn for placing ahead in a new line near the face.

Variable loads are carried by the dynamometer props, the load increasing in one case to about 17 tons 42 hr. after first setting, the corresponding subsidence being about 3 or 4 in. In another case the load increased to about 21 tons, 20 hr. after setting but with a subsidence of less than 2 inches.

The writer suggests that the figures of loading on the dynamometer props in the cases cited cannot be applied to the unit-area weight of the roof carried by the props in the vicinity because the dynamometer props were used singly and among lines of ordinary props, so it cannot be said whether the dynamometer prop in any particular case was bearing more or less load than adjacent props. Evidently it will require a group of dynamometer props to obtain true averages of unit-area weighting of the roof under any particular condition. It is further suggested that to obtain reliable averages it would probably need at least four parallel lines of dynamometer props extending at right angles from the face to the goave, or pack-wall, and the readings should be taken through several cycles of advance of face, including withdrawal of props behind and placing ahead at the face.

One of the most interesting tests reported by Dr. Winstanley is subsidence of roof measured in a heading driven ahead of a longwall face. He states that subsidence took place 70 ft. in advance of the face. This was in a coal bed 1950 ft. below the surface. However, it is not clear from the data whether the diminishing height of heading recorded at this distance ahead of the face might not have been due in part or wholly to the natural swelling of roof and floor, following the removal of coal in the driving of the heading.

Mr. Walker's paper[17] on "Bord and Pillar Whole Workings" (*i. e.*, workings in the solid coal) throws light on the matter of contraction of passageways following their being driven, although in a coal bed less than one-half the depth from the surface of that in which Dr. Winstanley made

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[16,17] Reference of footnote 15.

tests of subsidence ahead of a longwall face. The bord-and-pillar workings where Mr. Walker made his tests were in a Durham County Colliery, in a 4-ft. bed at a depth of 860 feet.

The "bordways" (equivalent to narrow rooms driven at right angles to coal "faces" or cleats) were 15 ft. wide and the "headways" or "walls" (cross-cuts driven on the "butts" or parallel to the face joints) also were 15 ft. wide, and were driven 99 ft. apart between centers thus forming coal pillars 84 ft. square. While driving the "bords" and "walls" the dynamometer props (the Wazau) were set (see Fig. 5) 4 or 5 ft. from the face of the "bord" heading, and continuous records of pressure on the prop were made while the place was driven to connection with a "wall." From an initial pressure in setting the dynamometer props of ¾ ton the pressure rose irregularly, as indicated in one diagram of a bord, to about 2 tons when the face had advanced in eight days about 25 ft.; the pressure then rose slowly to 2½ tons as it neared the "wall." In another diagram given in the Walker paper, for a "wall" heading, the pressure rose only to about

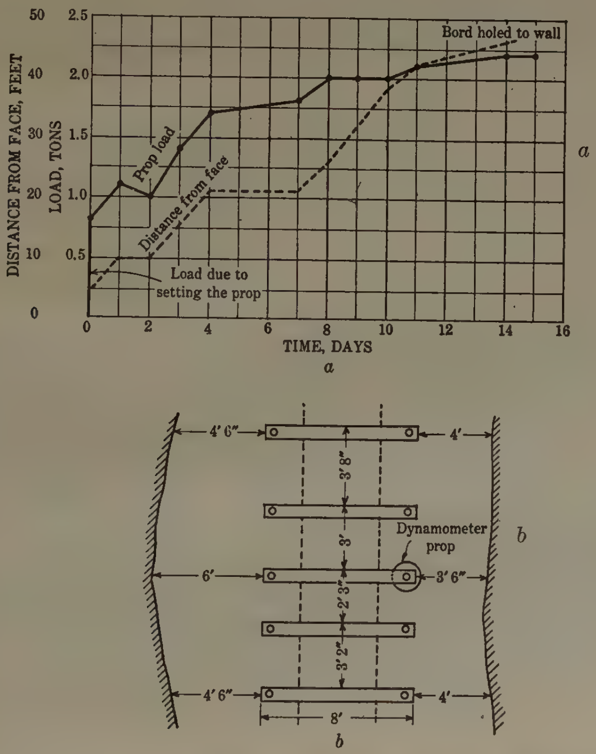

FIG. 5.—*a*, LOAD ON DYNAMOMETER PROP FOR DIFFERENT DISTANCES FROM FACE, DURHAM COLLIERY, ENGLAND. *b*, POSITION OF DYNAMOMETER PROP REFERRED TO ABOVE.

1½ tons with an advance of face in four days of about 25 ft., then the pressure fell off a little and was constant at 1½ tons. It seems probable that this falling off in pressure was due to floor penetration by the prop or a contraction of the wood plug in the top of the prop. It will be noted that the loads on the props were surprisingly light as compared with the maximum load on a prop in longwall workings referred to previously in connection with the Winstanley paper.

From the data reported it would appear that there is greater sag of a layer of the immediate roof in the bord headings (driven at right angles to the principal cleat joints) than there is in "walls" (butt headings) driven parallel with the principal joints or faces, which seems contrary to what might be expected.

Another important test is reported in Mr. Walker's paper: the boring of a vertical hole 4 ft. up into the roof to determine the subsidence of this higher layer of roof as compared with subsidence of the lower layer of the roof. The height reduction, in 21 days, of the upper stratum was found to be 0.29 in., while in the same period the height reduction of the lower layer was 0.55 in., indicating a separation of 0.26 in. By measuring with a scraper at the place of separation of the lower roof layer from the upper, it was found that the thickness of the lower layer was about 10½ in. This demonstrated that the load on the props and timbers was due to the weight of this lower layer only. It also seems probable that this layer, cantilevering out from the solid was partly self-supporting, which may also account for the small load carried by the props in the heading of which the dynamometer prop was one.

The temporary support of roof, by use of a movable hydraulic jack, while working on the open or goaf side of a pillar, with a special undercutting machine with very long cutter chain and in connection with a loading belt was a method developed about 10 years ago by Col. Edward O'Toole of the United States Coal & Coke Co., and employed in one of the mines of that company at Gary, W. Va. A roof-supporting device of similar nature termed a "hydraulic chock," has been constructed and experimentally used in the Ashington Colliery, Northumberland, England, and is described and illustrated in Safety in Mines Research Board *Paper* No. 61 (1930), page 55. The area of the roof bearing is 20 in. square, and it is mounted on a steel plate base with upward curved ends for skidding along on the floor. The jack is operated by a hand pump. The maximum roof load that had been carried is about 49 tons.

UNDERGROUND TESTING OF THE BEARING STRENGTH, ELASTICITY AND PLASTICITY OF COAL-IN-PLACE

The testing of the bearing power of coal-in-place is now being carried on by the Bureau of Mines in its Experimental Mine. The bearing power of soils and rock for foundation purposes has been tested in various ways, but as far as search of technical literature has disclosed, this is the first undertaking to test the strength of coal in place by known loads.

The initial problem arose through determining the strength of mine stoppings to resist an explosion pressure of not exceeding 50 lb. per sq. in. as specified in the Government coal leasing regulations of 1921, to provide for stoppings of that minimum strength to seal off mine openings to adjacent mines. Preliminary tests, made in cooperation with the Bureau of Standards, in a surface chamber at Washington having indicated that the most practical stopping for economical underground construction was a massive concrete slab acting as a flat arch, full-size stoppings of various thicknesses were successively erected in the Experimental Mine near Pittsburgh, Pa., and tested by explosions of specific strength as reported in Bureau of Mines *Bull.* 345.

In the course of testing it was discovered that strength of explosion-resisting concrete stoppings constructed as flat arches between coal ribs was dependent not only upon the compressive strength of the concrete but equally upon the deflection and resistance of the coal ribs acting as buttresses; the conditions of test being that no dependence should be placed on added strength to be obtained by contact with the roof or floor. Sole reliance on the lateral bearing strength of the coal was thought advisable for general application, as many mines have a weak roof and floor. Where the floor is strong, the stopping should extend down into a cut made in it. The roof, however, should not be cut whether weak or strong because it is likely to start planes of fracture, but if it is strong the stopping should be secured to it by placing anchor bolts in vertical holes and grouting the bolts in place. Where one or both of such measures can be done it adds materially to the strength-factor of safety of the stopping.

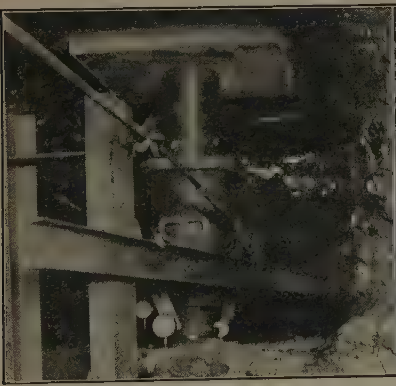

FIG. 6.

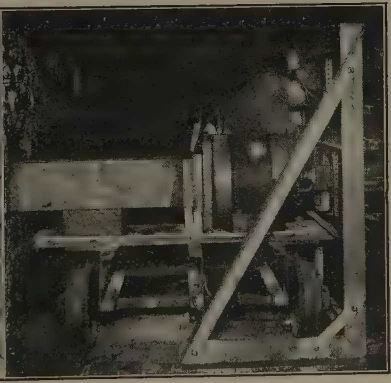

FIG. 7.

FIG. 6.—TESTING VERTICAL BEARING STRENGTH OF COAL WITH A 60-TON HYDRAULIC JACK IN THE EXPERIMENTAL MINE.

FIG. 7.—TESTING THE HORIZONTAL BEARING STRENGTH OF COAL WITH A 500-TON HYDRAULIC JACK IN THE EXPERIMENTAL MINE.

Testing the bearing strength of the coal horizontally and vertically was first undertaken by use of a 60-ton hydraulic jack as described in the appendix of the above-cited report. The arrangements as used for testing vertically are illustrated in Fig. 6. The vertical angle-posts, the upper part showing in the photograph, have footings in a block of concrete set into the floor and not in contact with the coal. The horizontal arm cantilevering from these posts is fastened only to them and is not held elsewhere. The purpose of the posts and arm is to hold Ames dial gages (which indicate strains) in contact with the coal; the dials of some of the gages show in the photograph.

The testing to determine the maximum bearing strength of coal-in-place within the limits of capacity of this jack, which required application of pressure to small areas only, developed some surprising results. The initial deflection under horizontal compression was not recovered completely on release of load and the unrecovered portion was termed "set." It manifestly was due to closing up of vertical cleat-joints of the coal. On further compression and subsequent decompression it was found that the coal acted elastically up to the point of crushing of the outer vertical layer of coal. On platting a graph of the curves of increasing unit-area pressure and then of decreasing pressure, the curves showed the characteristic hysteresis loop of a semi-elastic body.

On increasing the pressure beyond the maximum bearing strength, which the capacity of the jack limited to a bearing plate 2 in. square, a sharp report occurred just as in crushing a block in a compression machine and simultaneously the load on the jack was released through contraction of the coal.

Using the 2-in. square pressure plate, a maximum lateral bearing strength value of approximately 14,000 lb. per sq. in. was obtained. The crushing strength of the small cubes (2½ to 4 in.) of coal from this same seam, the Pittsburgh bed, has been found to have an average maximum strength of only 2500 lb.[18] Lawall's tests[19] on 3-in. cubes showed 5 to 30 per cent lower crushing strength parallel to bedding than perpendicular to it.

It was manifest from the tests on the stoppings that under momentary loading of an explosion pressure, the unit bearing strength of the coal-in-place as a buttress was greatly in excess of the unit compressive strength of the concrete as a column which by standard method of testing had a value of about 2500 lb. per sq. in., so it was not considered necessary to carry the explosion testing of stoppings further. It was deemed advisable, however, to obtain more information on the lateral strength of coal-in-place as buttresses for hydraulic dams, high pressure on which might be maintained indefinitely, instead of momentarily as in the case of an explosion pressure on a coal-mine stopping. It was also thought important to determine the vertical unit-bearing strength of coal-in-place as a foundation, for example, for a shaft or other mine constructions, or as a pillar to be left permanently for support of important surface structure or railway.

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[18] Reference of footnote 3.

[19] Reference of footnote 2.

It was evident, as indicated in *Bulletin* 345, that to determine satisfactorily the bearing power and deflection of coal-in-place laterally and vertically, pressure had to be applied to much larger areas than those which could be used with the 60-ton jack and accordingly the largest capacity hydraulic jack was desired that could be handled in mine workings. A 500-ton (1,000,000 lb.) pressure capacity hydraulic jack was obtained and is now being used in lateral compression tests with square areas up to 200 sq. in. (14.14 in. square) in size. The use of the larger pressure areas has developed some interesting phenomena in connection with the lateral bearing strength of coal-in-place. Fig. 7 shows a set-up in the Experimental Mine. The dial gages to indicate deflection and subsequent expansion of the surface of the coal surrounding the pressure plate may be seen between the supporting frame and the coal.

General Indications Inferred from Effects in Use of Larger Jack

The testing so far has been confined to lateral pressures on the coal-in-place. This work is being carried on by H. P. Greenwald and S. Avins, physicists of the Bureau of Mines, and a progress report, to be published by the Bureau of Mines, is in preparation. Certain preliminary indications based on the testing in the Pittsburgh coal bed have been summarized by the writer, who has also added some conjectures as to physical phenomena not yet proved of either the Pittsburgh coal or other coals in place, respecting the lateral bearing strength and deflection of coal-in-place. They are as follows:

1. That coal-in-place tested laterally up to certain limiting pressures may be classed as semibrittle material and has definite elasticity.
2. That the coal when pressed deflects a certain amount which is not recovered when the load is released as already found in previous testing with small plates, this limited deflection being due to closing joint spaces. Then with increasing pressure further deflection is due to the elasticity of the coal substance which has the side restraint from adjacent coal.
3. Pittsburgh coal between vertical jointings (butt and face joints) pressed by plates whose area is not over 4 in. square gives an ultimate bearing strength of 10,700 to 14,000 lb.; when rupture of the surface (vertical) layer of coal occurs there is a sharp report.
4. When compression is by successively larger plates the sudden rupture accompanying failure gradually disappears. When the area pressed is 200 sq. in. a condition of plasticity is indicated by the continued deflection under increasing load with displacement of the coal surrounding

the area pressed. Moreover, the rate of deflection under a specific load has a time factor; *i. e.*, if the jack is not pumped the load falls. This factor will be determined in further testing by employing a constant load.

5. The face of the coal for a zone of a foot or two or more around the plate as the pressure is applied is deflected inward (which is indicated by the gages). As the load is increased, the outer vertical layer of coal shears around the edges of the plate, and the deflection still continues but is less than that of the coal under the plate; apparently it is the layer deeper than where the shearing penetrates that carries, by its deflection as a beam, the outer layer with it. The rate of deflection of surrounding coal decreases as bearing strength is approached and finally becomes zero, following which its direction of movement is reversed and is in opposition to the direction of applied pressure.

6. If the bearing strength of the coal-en-masse be considered to be the point of elastic limit beyond which permanent deformation or flow of the coal takes place, then the results of the tests indicate that the larger the area pressed the less the unit bearing strength. Apparently the bearing strength value is determined by some relation between the total pressure and the area of the loading.

7. For the conditions of test it is indicated that lateral bearing strength of coal-in-place for plates from a size of 5 in. square to the largest used, 200 sq. in., varies from 6500 down to 5000 lb. per sq. in., the curve of bearing strength indicating a possible minimum of 4250 pounds.

8. The deflection of the surface of the coal directly under the pressure plate or in effect, the movement of the plate under increasing load, was at a pressure of 4000 lb. per sq. in., of the order of about 0.45 in. for an area of 200 sq. in. This deflection includes the closing of the open joints behind the area pressed in addition to that due to elastic property of the coal substance.

9. While a plastic condition of the Pittsburgh coal was indicated by the tests with larger plate areas, it is a question yet to be determined whether application of pressure on an area of coal within a deep recess cut into it, as for the buttress of a hydraulic dam, would not give a higher unit-area bearing value because of the restraint of the coal on either side of the cut.

10. It is conjectured by the writer that when lateral constant pressure on coal is applied on the face of a cut made into the solid coal and the pressure is sufficient to cause the coal under the area pressed to be plastic and to start flowing, the flow will continue until the joint spaces within the spreading lines of stress are more or less filled. But there would be constantly increasing resistance, frictional and deformational, so that the ultimate bearing load would rise to a higher value than indicated by pressing the open surface of the coal. It seems possible that the

limiting condition would be the restraint offered to deformation and flow by the coal ribs on either side of the cut and in a coal bed of uniform physical strength, the restraint may bear some determinable ratio to the depth of the cut.

Concluding Remarks

The foregoing descriptions and discussions of the methods of research relating to underground rock and coal stresses and movements in coal mines, being carried on in different countries, include all the recent fundamental research work that has been brought to the attention of the writer. It should be again emphasized, as stated in the beginning, that this paper embraces only studies of roof subsidence and underground movements of the immediate overlying rocks in coal mines and does not include studies of caving or subsidence of the surface.

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